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Sintering Fine Iron-Bearing Materials by Blast-Roasting Machines.

Mr. James Gailey, who needs no introduction to our readers, has presented an exceedingly interesting paper before the June meeting of the American Institute of Mining Engineers on the sintering of flue dust on the Dwight-Lloyd system. As is well known this machine consists of a moving train of perforated iron pallets passing over a suction-box and that on this is placed the ore which is desulphurized by the blast principle of roasting. The machine resembles in appearance a short belt-conveyor.

This machine has been used successfully for the roasting of lead and copper ores. Mr. Gailey describes some experiments looking toward its application in the sintering of flue-dust and ore from the recently discovered mines of Mayari, Cuba. This proposal is to replace the sintering of the ore in long nodulizing kilns. The experiments described are favorable to the proposal. Both ores were delivered from the sintering machine with a large proportion that was coarse or granular. The percentage of iron was increased in the case of the Mayari ore from 48.53 per cent to 53.97 per cent and in the case of the flue-dust from 46.06 per cent to 57.90 per cent. The first increase is mainly due to the burning out of the coke while the second is due to the dehydration of the ore. All the pieces of the ore are described to have consumed a cellular structure like coke which is peculiarly desirable for the blast-furnace.

Experiments were made with magnetic concentrates and it is remarkable to note that the sulphur was reduced in the sintering from 1.17 per cent to 0.006 per cent. In the case of the magnetic concentrates and the Cuban ore, seven per cent of coal (presumably bituminous coal) was added. The flue-dust contained enough coke to furnish heat by its combustion for the operation. Mr. Gailey estimates the power needed for a 100-ton plant at 45 hp., the cost of repairs at 6 cents per ton. The capital cost of plant is low, as a 100-ton plant only weighs 26 tons.

We believe that this is a most important common-sense application of a machine devised for one operation to a process somewhat different in purpose but similar in modus operandi. The question of flue-dust in the smelting of Mesabi ores is most important. If the ore could be screened and the objectionable fines eliminated and sintered in a cheap efficient process a lot of blast-furnace managers would sleep better. Though we do not hear much now about blowing off the tops of blast-furnaces, yet Mesabi ores are smelted by slips and falls and the smelting is hard on the nerves.

It is very apparent from this and other proposals that the metallurgical engineer who recently proposed to amortize his plant on the ground that it would not be obsolete for twenty years was far from right. "The depreciation due to

the advance in the state of the art," should be at least ten per cent. The metallurgical profession has not given this important point the due weight that the electrical profession has.

Arbitration of the Smelter Fume Question.

The recent announcement of a suit brought by ranching interests in Nevada to restrain smelting operations on the part of the Mason Valley Mines Company, at its plant which is now in course of construction at Thompson, Nev., brings forcibly to mind the necessity for devising some means of reconciling the conflicting interests of agriculture and metallurgy. Both are of fundamental importance as original sources of wealth, and as such their interests are in a measure mutual and interdependent, especially when their several operations are carried on in the same locality. No permanent good can come from suits or injunctions, nor from resort to warfare of any sort. The settlement of the problem requires calm and impartial investigation, consideration of the interests of both parties, and, above all, extreme fairness in placing restrictions on the one or granting indemnity to the other.

That smelter fume may be and has been the direct cause of damage to vegetable life does not admit of contradiction. Three principal sources of damage are recognized, viz., sulphur dioxide, sulphur trioxide and solid fume such as arsenic. The best scientific talent in the profession has been marshalled to study the nature and extent of the damage which these substances cause and the possibility of taking reasonable steps to prevent their indiscriminate discharge into the air. It has been pretty thoroughly proved that it is possible to prevent the discharge of sulphur trioxide and solid fume, but it may be asserted that in the present state of the art no reasonable and effective method has been proposed to prevent large quantities of sulphur dioxide from escaping into the air and doing possible damage.

Sulphur trioxide may be neutralized with zinc oxide and collected in bag houses as at Midvale, Utah, and Kennet, Cal. Solid fume may be settled in flues by cooling to the proper degree of temperature and causing it to impinge on baffles as at Great Falls, Mont., and other places. But sulphur dioxide presents a different problem. It is not so much the fault of chemical knowledge that this gas is not handled as well as sulphur trioxide, but rather the difficulties encountered on the mechanical and commercial side. Many proposals have been made which undoubtedly will neutralize this gas and eliminate it as a source of damage. But what about the cost of equipment and operation of these schemes? How about the difficulty of handling large quantities of flue gas without seriously interfering with the draft on the furnaces? What shall be done with some of the neutral products obtained by these various proposals, and finally, if sulphuric acid be made from the gas, what shall be done with the acid?

Here is a problem which any sane farmer must not be too insistent in forcing the smelters to solve. The latter are fully conversant with the situation, much more so than the former, and, we are proud to say, the spirit of fairness and generosity has marked the attitude of the smelters more than that of the farmers. The larger companies, against which most of the agitation has been directed, acknowledge the possibility of damage and realize their present limitations in preventing it. It is a matter of economy with them to prevent the indiscriminate discharge of fume. Furthermore, they stand ready to pay for unavoidable damage and regard such payment as one of the expenses incident to the business; but they want the damage fairly assessed with due regard to all the conditions.

There are things which can be settled only by a fight—by a fight to a finish. It seems almost foolish to deny this. But can any sane man think seriously of a fight to a finish between agricultural and metallurgical interests? We believe that the solution of their problem lies in arbitration and that arbitration will prove in this restricted field immensely more successful than in the evasive and spectacular field of international politics. We decry suits and injunctions as being only so many expensive obstacles in the way of progress. The Anaconda Copper Mining Company and the United States government have set a good example in appointing a technical commission to consider the question as far as it relates to their several interests. This commission has arbitrary power to decide whether material damage is done to the government, and what steps, if any, the Anaconda company should take to prevent such damage. The arrangement is eminently satisfactory and is regarded on all sides as the only fair solution of the problem and one which should be adopted instead of resorting to the courts. Suits are expensive and leave one party dissatisfied. Injunctions only paralyze industry and frequently reflect injury on those who obtain them. Arbitration will be cheaper in every sense of the word, will possibly satisfy all concerned, and will surely reflect credit on the intelligence of both parties to the controversy.

The Metal Markets.

As we have pointed out several times, the increase of the productive capacity of the various metallurgical plants of the country in the past ten years was so rapid as to overrun the consumptive capacity of the country. From 1900 to 1907 there was almost without halt an increase of from 7 per cent to 10 per cent per year in the production of iron, steel, lead, copper, zinc, and the minor metals and this increase was compounded. At the present time we believe that these several basic industries are recovering from the period of mental unsettlement that occurred after the period from June, 1909, to July, 1910, when the wheels of commerce seemed to be rushing again. At present the production is going directly into consumption. As this consumption is on a hand-to-mouth basis, it is hard to see how it can be much reduced. As this situation is quite widespread throughout all industries, it would seem that the metal prices have about touched rock-bottom and the general tendency in all should be upward. There would be little doubt about this upward tendency were it not that the volume of all crops except that of cotton seems bound to be below the average of the past four years.

Sober-minded people while they may not exactly prefer a period of liquidation and retrenchment do not dread it, for that is the period when everybody works more and spends less, and when the entire spirit of inflation and speculation is curbed and chastened. This conservative policy of producers and the everywhere manifest attempt to reduce fixed charges,

to eliminate useless expenses, to improve the efficiency of plant and workmen, has put the metal industry in good shape to meet any disappointment that can result from the diminished harvests.

It also can be remarked that there are a great number of far-seeing and able inventors in this country and that their endeavors are appreciated more in periods when it is a question of a maximum of plant efficiency rather than a question of a maximum of plant capacity. Their efforts will reap a golden harvest when rightly and conservatively employed. Furthermore keen observers notice among the plain people of the United States a manifest displeasure with the attempts at "trustbusting" simply for the glory or fun that may be in it and a desire for a sane regulation of these instruments of production. When all these curative influences have done their work, we shall see a more normal growth of the metal industry. In the immediate future the outlook is not particularly promising.

Western Hospitality.

After a red-hot summer day in Omaha, to wake up the next morning in the Pullman sleeper in Colorado and see through the windows in the distance the snow-clad peaks of the Rocky Mountains, is a sensation which has given a thrill to many a traveler from the East. The stately range of peaks in the glory of their eternal snow offers the first greeting to the stranger who feels that he is entering God's country. And when on leaving the train in Denver he turns toward the city, there is the big "Welcome" Arch—that pretty and touching symbol of Western hospitality.

It has been truly said that nobody who has not seen the West and the Pacific Coast can have a full and proper conception of this country and its people. It is true, the East is mighty, and New York City is really a wonderful sight with its feverish commercial activity concentrated in such a limited space that its buildings must grow to scrape the sky. Yet the East and all its activity would be unthinkable if it were not backed up by the unlimited natural resources of our big and beautiful West. There is everything big and great in the West—the giant trees in the virgin forests, the orange groves, the wheatfields, the mines, the concentrators, the smelters. And the very men themselves—generous, cheerful, big-hearted, enthusiastic, and of a never-dying optimism. How could they have built up again a new and more beautiful San Francisco after the calamity of only a few years ago, if it was not so? Nature, bountiful and generous, is reflected in the big-hearted optimistic generosity of those empire builders of the West. And its natural expression is Western hospitality which no one who once enjoyed it can ever forget.

The Western man, whether he be a metallurgist or a mining man or anything else, always gives himself as he is. He is not afraid to show his smelter or concentrating plant to the interested visitor. There is nothing about him of that anxious secretiveness which is such a distinguishing trait of some of our Eastern and European friends. While his work speaks for him, he is not afraid to speak of his failures. He believes in the do ut des. He is always an optimist, and where old methods fail he looks out for new ones, and extends to them his hospitality.

In some parts of the West where the immense strides in agriculture have been coincident with a temporary decline in mining, the latter industry has suffered a loss of prestige as a money-making business. This has been due partly to a lack of spectacular discovery or development in new mines, and the cry has gone forth that new Cripple Creeks must be found in order to stimulate the industry. That the old spirit is still alive is shown in a broader interest and greater activity in prospecting and development, more evident now than has been apparent for some years past, and undoubtedly some good will result. But in another direction and in a far more important manner the true Western pioneer spirit shows itself now in the willingness to accept and attack new methods of metallurgical treatment.

Indeed, of greater importance than the discovery of new and rich mines, is the development of improved metallurgy for the treatment of ore already known to exist and, in some cases, already mined. As a measure of conservation of our natural resources improved metallurgy is of fundamental importance and we may look to see greater results accomplished in the immediate future by solving metallurgical problems than from the discovery of new mining districts. The most active metal-mining districts in the West to-day are those in which the metallurgical problem has been solved. The "dead camp" is the one that abounds in ore that cannot be treated at a profit. The main point is that the West is fully alive to the problem of the situation, and that means the West will solve it by its own energy and by its willingness to try new methods even if not yet completely proven. The Noble electric iron ore reduction and steel plant, now going up in California, is a true expression of this new metallurgical pioneer spirit.

In general it is evident that the limits for successful ore treatment are gradually widening. Not that any new and startling process is proving a panacea for all troubles, but accessory devices and schemes applied here and there in conjunction with well known and proved processes are raising the percentage of extraction and extending the application of the processes to a wider field. It is a fact that an old slag dump of a smelter near Denver which was dismantled not so long ago is now being treated for its metal content at a good profit.

In concentration where great waste has hitherto existed, improvement is being made by the use of flotation as an accessory process. Dry chlorination holds out some hope for the treatment of complex ores; in two western plants it will soon have an excellent chance to show what is in it of commercial success. Cyanidation is being extended by the aid of additional chemicals; and from physical chemistry the phase rule is borrowed to base on it an interesting modification of cyanide practice (as described in the suggestive articles of Mr. Lowden and Mr. Rothwell in our July issue and in this issue respectively). Electrolytic schemes are in sight which may prove of value in conjunction with present processes, while the electric furnace is anxiously awaited as one means of smelting zinc ores carrying other metals. In short, Western metallurgy is ready to make use of all latest advances in chemistry, electrochemistry, and physical chemistry. The outlook is very bright and this journal will rejoice in recording how this new phase in the evolution of Western metallurgy is carried through to success by Western hospitality.

Toronto Meeting of the American Electrochemical Society.

The twentieth general meeting of the American Electrochemical Society will be held in Toronto, Ontario, Canada, from September 21 to 23 (Thursday to Saturday).

Sessions for the reading and discussion of papers will be held on the morning and afternoon of Thursday, while a smoker with entertainment by section Q is scheduled for Thursday evening.

On Friday morning there will be a session for reading and discussion of papers, probably at some golf club. In the afternoon a golf tournament, etc., will be held, and in the evening a banquet at McConkey's restaurant.

On Saturday morning an excursion will be made to the Canadian General Electric Company's plant at Peterboro, with the return trip through the locks to reach town in time to catch the 5 p. m. boat out of town.

The full program of papers will be published in our next issue. Members having further papers to present, should notify before Sept. 1st either the chairman of the committee on papers, Mr. F. A. Lidbury, Oldbury Electrochemical Company, Niagara Falls, N. Y., or the Secretary of the Society, Prof. Jos. W. Richards, Lehigh University, South Bethlehem, Pa.

The local committee for the Toronto meeting consists of the following gentlemen: Dr. W. Lash Miller, Chairman; Wallace P. Cohoe, Chas. Danforth, Saul Dushman, Thos. C. Irving, Jr., J. A. Kaemmerer, Peter Kirkegaard, Jas. W. Moffat, L. V. Redman, Thos. D. Robertson, A. O. Tate, Geo W. Watts, O. H. Wurster, and any Niagara Falls members as is deemed advisiable.

The Iron and Steel Market.

While August has been rather a dull month as to the placing of new business in iron and steel, and has seen considerable wavering of prices, the trend of production has been slightly upward on account of the fairly heavy buying in the latter part of July and on the whole the developments of the month were reassuring, for the expectation was that this year July and August would prove very unsatisfactory. It may be well to observe that the iron and steel trade in the past two months has not been in as good condition as has been widely reported. Unusual efforts have, indeed, been made in some quarters to spread the impression that the iron trade has been rapidly improving since about June 1, when as a matter of fact there has been little real improvement and the principal cause for congratulation is that what was expected to be a very trying period has been passed without serious trouble. Production, as measured by pig iron, has decreased month by month since March, whereby production in July was at the rate of about 21,500,000 tons a year, about 4,000,000 tons less than the April rate. Late in July idle furnaces began to be put in blast and this increase in number has continued through August. On account of the high humidity always prevalent in these months, the additional furnaces have not served greatly to increase production, and indeed there is reason to infer that they were put in largely to maintain, rather than to increase, the rate of output. Under the system of individual ownership which formerly prevailed, production almost invariably suffered a large decline in the midsummer months, because individual interests could not afford to blow furnaces in and out by reason of such a temporary influence. With the United States Steel Corporation the case is different. Having so many furnaces, it always has a few which are about ready to go out, while it has others which have been relined and can readily be blown in. There is no doubt, however, that August has produced somewhat more iron than July, and it is certain that stocks, both merchant stocks and those of steel works, have been drawn upon at a fairly heavy rate. The steel works stocks are now very light, while the merchant stocks, although decreasing somewhat, are still very heavy.

Prices of finished steel products evinced more irregularity in August than in July, although there were important irregularities in the former month. Practically all the important finished products except rails have been subject to cutting, and indeed there is little semblance left of the controlled market which existed for so long.

An important feature of the market as now made is that the cutting usually proceeds by very small increments. There is no slashing of prices, but rather the order is frequently obtained by a slight concession, a relatively small fraction of a dollar per ton. A definite effort is being made by selling departments to shake off the habit of deep cutting and to substitute the method which has always prevailed in the foreign market of quoting prices very closely, allowing the fraction of a shilling or mark per ton to turn the scale. Another influence is that the cost level has been rather closely approached by the successive declines which have occurred since the market reached a top point late in 1909.

Steel bars for prompt shipment are being sold regularly at 1.20 cents, Pittsburgh, when the specifications are at all desirable, and this price is sometimes shaded slightly on the most desirable orders, the regular or nominal market being 1.25 cents. Plates are being shaded regularly $1 a ton from the regular price of 1.35 cents, and occasionally the lower figure is shaded. Structural shapes are fairly firm, 1.35 cents being only occasionally shaded. Sheets are being shaded $1 a ton in most districts, and in Ohio and Chicago territory are frequently shaded $2 a ton. Tin plates can be purchased at 10 cents per box concession for early delivery.

The mills insist that little if any forward business is being booked at cut prices, and that as order books fill up the tendency to cut will gradually disappear, although it is presumed that as the effort is made to withdraw cut prices a spell of particularly dull business may be experienced. The difficulty is that the mills are not satisfied with running somewhat heavier than a month or two ago, but are individually endeavoring to work up to full operation, and that buyers will not purchase for forward delivery as long as they see that prompt material can be purchased at a concession, so that it is far from evident that the market is really lining up for an actual restoration of the nominal quotations as the real trading basis.

Pig Iron.

Nearly all the Southern producers have withdrawn the price of $10, Birmingham, and are quoting all the way from $10.25 to $11, but there is still a little iron to be picked up at the $10 figure for early delivery, and consumers show no disposition to pay any advance, so that Southern iron has been very quiet. In the North prices are not materially changed, except that in eastern Pennsylvania basic has advanced about 50 cents. We quote: f. o. b. Birmingham, $10 to $11 for No. 2 Foundry; delivered Philadelphia; basic, $14.75 No. 2 x, $15, f. o. b. valley, 90 cents higher delivered Pittsburgh; Bessemer, $15; basic, $13; No. 2 foundry, $13.50; malleable, $13.25 to $13.50.

Steel.

There has been some shading of regular billet and sheet bar prices, which are $21 for billets and $22 for sheet bars, f. o. b. maker's mill, Pittsburgh or Youngstown. Open market sales, however, have been light, consumers being well covered by contracts. Rods are $27, Pittsburgh.

Finished Products.

Regular prices on finished steel, f. o. b. Pittsburgh, except where otherwise stated, are as follows, but are subject to considerable shading, as already noted:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.35 cents for tank quality.

Shapes, 1.35 cents for beams and channels, 3 to 15-inch, inclusive, zees, and angles 3x3 and larger.

Steel bars, 1.25 cents, base; iron bars, 1.25 cents to 1.30 cents, Pittsburgh or Youngstown; 1.27 cents to 1.30 cents, Philadelphia, and 1.20 cents to 1.25 cents, Chicago.

Wire nails, $1.70, base plain wire, 1.50 cents, base.

Black sheets, 28-gage, 2 cents; galvanized, 3 cents; blue annealed, 10-gage, 1.50 cents; painted corrugated roofing, $1.40 per square; galvanized, $2.55.

Tin plates, $3.70 per base box.

The Western Metallurgical Field

Indexing Metallurgical Literature.

Engineers who appreciate the value of indexing the rapidly growing literature on metallurgical subjects will welcome the work recently published in the April, 1911, *Quarterly* of the Colorado School of Mines. The article is by Robert M. Keeney, Fellow in Metallurgy at the School, and consists of an extension of the Dewey decimal system of classification applied to metallurgy, metallography and assaying. The work is exhaustive and gives a complete system of indexing the subjects mentioned.

In the Dewey system the entire field of knowledge is divided into ten principal divisions, numbered from 0 to 9. These are, 0, General Works; 1, Philosophy; 2, Religion; 3 Sociology; 4, Philology; 5, Natural Science; 6, Useful Arts; 7, Fine Arts; 8, Literature; 9, History. Each of these main divisions is further subdivided into ten sections, and the process of decimal division is repeated as often as found necessary. This makes it possible to represent the proper classification of an article or subject by means of figures as shown in the following example: Thus, 669 means Class 6 (Useful Arts), Division 6 (Chemical Technology), and Section 9 (Metallurgy). In carrying out the system, a decimal point is placed for convenience after every third figure. To illustrate further, we have 669.145.412 (Bessemer Converter Tuyeres); 669 as already explained means Metallurgy; 669.1 the metallurgy of iron and steel; 669.14 the manufacture of steel; 669.145 the Bessemer process for manufacture of steel; 669.145.4 the Bessemer converter; 669.145.41 the construction of the Bessemer converter; and 669.145.412 Bessemer converter tuyeres.

While it is clearly impossible to include all the terms used in metallurgy, metallography and assaying, enough can be given to enable the user to assign the proper number to almost any subject that may arise. The decimal system of classification is adapted to the indexing of books, notes, clippings from journals, drawings and catalogs. In indexing current literature where the number of cards becomes large, it will be found convenient to keep the card index in order of the numbers of the decimal system rather than by alphabetical order.

Engineers can use the system in three ways: First, to keep a card index of technical literature; second, to file clippings from journals, notes, drawings and catalogs, without a card index; or third, for a combination of the card index with the filing system, so that a reference to a journal from which there is no clipping at hand can be recorded with the other references. In filing clippings, an envelope 7¾ inches long by 3¼ inches wide is a convenient size.

Mexican Labor Troubles.

Following close on the heels of the revolution, which seriously hampered mining and metallurgical operations in Mexico for several months, comes a period of unrest in labor circles which also threatens to impede progress for some time. Native peon labor is dissatisfied with several conditions under which it works. First, wages are believed to be too low, and an increase of from 25 to 100 per cent is asked from the various mining and smelting companies. The peon wage ranges from one to two pesos per day. Secondly, there is objection to some arbitrary deductions which are made from wages to cover medical attendance and hospital fees. This has been an item objected to in many other places, so that it is not surprising to find it prominent in the present Mexican troubles.

The northern part of the republic is most affected. In the state of Chihuahua there are about half a dozen camps in which serious labor strikes have occurred. The Chihuahua plant of the American Smelting & Refining Company has been closed since June 27 on account of a strike among the peons. The shut-down is apparently for an indefinite period, as the company gave its American employes thirty days' notice of discharge. The camp of Parral, Santa Barbara and Santa Eulalia also are threatened with serious strikes, and as there are many important mines and mills at these centers, the outlook is not encouraging. The opinion exists among those conversant with the country, that no definite settlement of labor conditions can be expected until after the national election in the coming October. The present dissension is not altogether unlooked for as a natural consequence of the recent revolution, and the control of the situation will be one of the problems of the reconstruction period. No doubt professional agitators will try to take advantage of the situation, but it is hoped for the good of the country and the mining industry that peaceful industrial conditions will be reached this year.

Spelter Statistics for 1911.

Statistics compiled by the United States Geological Survey show that the production of spelter or metallic zinc from ore for the first six months of 1911 was 140,196 short tons, a gain of more than 5,000 tons over half the record production of 1910. Of this production, 5,135 tons was made from foreign ore. Spelter stocks were reduced from 23,232 tons to 17,788 tons. Imports remained about the same, but exports were nearly double those of half the preceding year. The apparent consumption of spelter was 135,497 tons, an increase of more than 12,000 tons over half of 1910, but about the same as in half of 1909. The average price of spelter at St. Louis for the period was 5.36 cents per pound, the London average being 0.2 cent less per pound. During the latter part of May and the first of June the average London price was about 0.1 cent higher than the corresponding St. Louis prices. Under this stimulus the May exports of spelter, zinc ore, and dross were largely increased over those of the preceding months.

Camp Bird, Ltd.

This company, which owns the Camp Bird mine in Colorado and the Santa Gertrudis mine in Mexico, has recently made some important changes in its financial policy. It appears that at the time of the purchase of the Santa Gertrudis, in the latter part of 1909, it was necessary to raise between four and five millions of dollars rather quickly or lose the opportunity to purchase the property. In order to do this the company issued debenture stock which it was to redeem at the rate of about $500,000 per annum, with interest amounting to about $150,000 per annum. Although the terms of this issue were rather stringent, nevertheless the directors felt that the mine was too good to lose, and acceded to the terms.

Later developments in the Santa Gertrudis showed that the mine promised far greater profits than were at first realized, and the directors felt that it was too valuable a property to be subject to the mortgage represented by the debentures, and that the company should take steps to redeem the debentures at once and remove the risk of losing the property through some unfortunate circumstance during the next five years. It was therefore proposed to increase the capital of the company by the issue of 750,000 shares of one pound sterling each, being preferential shares drawing 7 per cent, and sharing with the ordinary shares in any surplus remaining after paying the usual 20 per cent on the ordinary shares. By this arrangement the company would remove the possibility of losing a valuable property for a small mortgage before the debentures were paid

in full in 1917, and the only expenditure which would take precedence over dividends on the ordinary shares would be the interest on the new stock, amounting to about $225,000, instead of $650,000 as under the present arrangement.

At a meeting of the stockholders, held in London on July 18, 1911, a majority interest voted to increase the capital as above outlined.

The latest reports from the company's properties are to the effect that the Camp Bird reserves have been increased by substantial bodies of ore, and that on July 1 there was a further profit of about $350,000 in sight. Profits are expected at the rate of about $50,000 per month for the next six months. At the Santa Gertrudis the company has developed about 1,200,000 tons of ore, which is estimated to give a net profit of nearly $6,000,000. In addition to the ore blocked out there is a large tonnage of low grade ore left in the mine as filling. This will be milled with the high grade ore.

Spelter Improvements.

It is reported that the American Smelting and Refining Company is to make improvements at the East Helena, Montana, plant, which will cost about $75,000. The lead smelting business in the West is generally in a poorer condition than in years gone by, and at the same time the competition in some states is keener. The recent entrance of the International Company into the field has stimulated the business somewhat, and other companies will have to meet the competition of a new and well equipped plant. It has been reported before that the East Helena plant was to be closed, but the recent decision to improve it effectually disposes of this rumor.

Improved Market for Lead Ores.

The entrance of the International Smelting Company into the field of lead smelting is expected to stimulate the production of that metal by reason of the reduced rates at which lead ores will be treated. Recently several good contracts have been obtained by this company, and it is certain that they have been secured only at rates advantageous to the shipping companies. The output of the Hecla Mining Company, in the Coeur D'Alene district, Idaho, is to go to Tooele for the next five years. This ore formerly went to the Ohio & Colorado plant at Salida, Col. The entire ore and concentrate production of the Monarch mine, near Murray, Idaho, also has been contracted to the International company. These two sources of lead are worth a great deal to any company engaged in custom lead smelting, as the product is steady and of good quality. The effect of the activity of the International company's agents is to place the lead producer in a more favored position than he has enjoyed for some time past.

CORRESPONDENCE

Fixation of Atmospheric Nitrogen.

To the Editor of Metallurgical and Chemical Engineering:

Sir.—I should be glad to know how Mr. John Woods Beckman can write (Vol. IX, No. 7, page 340): "Recently the Birkeland and Eyde process has been abandoned and the Schönherr process of the Badische Anilin Fabrik has been installed and is extensively operated at Notodden, Norway, where the company's factory is located."

Not a single furnace of Schönherr's system is burning at Notodden, but 35,000 kw is operated there in Birkeland-Eyde furnaces.

University of Kristiania, Norway. Kr. Birkeland.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir.—Your July number contains a letter by Mr. John Woods Beckman, entitled, "The Manufacture of Air Salpeter, a Well-Paying Industry."

While the article as a whole is quite correct, the author is misinformed on some points as regards our company, and as we are the owners of the Birkeland-Eyde patents for the nitrogen fixation process and of the only nitrate manufacturing plant existing in Norway today, we beg to make the following statements:

The Birkeland-Eyde process has not been abandoned; on the contrary, the entire plant at Notodden is employing the Birkeland-Eyde process exclusively.

Our company is capitalized at Francs 41,166,250 of which Francs 4,500,360 is preferred and the remainder common stock.

Norsk Hydro-Elektrisk Kvælstofaktieselskab.
Kristiania, Norway.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir.—With respect to the third paragraph of the letter of Mr. J. W. Beckman in your July issue I wish to make the following correction:

The several electrical furnaces of the Notodden plant which were formerly of the Schönherr (Badische Anilin und Sodafabrik type) have all been replaced recently by the furnaces of the Birkeland-Eyde (Société Norvégienne de l'Azote et Forces Hydro-Electriques) type.

In the large plant at Rjukan to be built jointly by the above mentioned firms, furnaces of both types will be installed in order to definitely determine which of the two types gives the best results in commercial service.

With reference to the chemical part of the Rjukan plant it had been decided some time ago to use the absorption towers of the Badische Company. It has been decided recently, however, that absorption towers of the Birkeland and Eyde type will be used at the Notodden plant.

It may be of interest to remark that the Notodden works, of which the present capacity is from 50,000 to 60,000 horsepower, belong entirely to the Société Norvégienne de l'Azote et Forces Hydro-Electriques, using and exploiting the Birkeland & Eyde patents.

Paris, France. A. S. Garfield.

Continuous Cyanidation.

To the Editor Metallurgical and Chemical Engineering:

Sir.—I have read with great interest Mr. Rothwell's valuable article in your issue of July, 1911, in which he suggests a continuous treatment method of cyanidation. The method he describes so clearly consists of all sliming, continuous agitation—which is apparently appreciated by all cyanide men since its introduction at the Esperanza last year—and continuous decantation.

It will probably interest Mr. Rothwell very much, and also your readers, to know that the essential features of the latter were first used in practice, I think, about nine years ago by Mr. John Randall at the Highland Chief Mill near Deadwood, S. D., and that several plants are now in operation along the same lines.

Mr. Randall grasped the most important point, which is the passing of the slime and its successive use for the dilution of the increasingly rich pulp as the solution passes toward the head of the series. This makes it necessary to precipitate the solution only after it has been several times enriched, thus reducing the amount to be precipitated to a small fraction of that otherwise required. He operated, as did the Denny brothers about the same time in South Africa, with large settling cones and attained fair technical results, although troubled with building up of solids on the sides of the cones, until the property was shut down for lack of ore.

It is unfortunate that Mr. Randall did not publish his results. The only reference I have seen to this is in a review of cyanide progress, by R. W. Gordon, in your issue of January, 1911, in which Mr. Randall is given credit for pioneer work along this line.

The advent of the Dorr thickener, which rendered continuous settling to a very thick pulp feasible, made the matter more

attractive and we suggested it to friends in 1906, but through pressure of other work it was not taken up seriously until 1910, when two plants— one to recover the values entirely by decantation, the other to use continuous decantation to reduce the values to a low point ahead of an Oliver filter—were installed and are now in successful operation.

A number of articles have appeared on the subject in the last year or two, though many writers, such as Mr. F. R. McCann, in *Transactions* of the Mexican Institute of Mining and Metallurgy, July 1909, while appreciating the advantage of continuous decanting, failed to grasp the idea of the reverse movement of the barren solution and so were perplexed as to the disposal of accumulated low-grade solution.

A short article by L. B. Eames, which appeared in the *Mexican Mining Journal* for September, 1910, under the rather misleading title of "Discussion," and which for that reason has probably not attracted the notice it merits, gives a very clear description of "Continuous Decantation." Besides a flow sheet, Mr. Eames has given an algebraic method for determining, from assumed conditions, exactly what the mechanical losses should be, which will be greatly appreciated by any one contemplating the use of this method.

Mr. Rothwell's flow sheet, covering as it does almost all conditions likely to be met in practice, is worked out so well that the cyaniding fraternity will find it of great value. I would call attention to one point, however, and that is that in the flow sheet given if the strength of solution used was two lb. or over, the mechanical loss in gold and cyanide would be less if the barren solution was added to the thickener next to the last. This loss would, of course, be further reduced by the use of cyanide solution instead of water in the batteries.

The remarkable percentage of recovery of dissolved values with the precipitation of a relatively small amount of solution, which should be obtained by this method, is well shown by the following figures, calculated from the conditions assumed:

1. Value per ton assumed to be dissolved from the ore before the pulp reaches the decantation system	$4.00	$4.00	$4.00	$8.00	$8.00	$8.00
2. Moisture in pulp discharge from thickener, per cent	37.5	50	50	50	50	50
3. Tons solution precipitated to $.02 per ton of ore	1.8	3.	4.	3.	4.	3.
4. Total thickeners used, the underflow into the last being diluted only with an equal amount of water to secure the minimum loss of cyanide	4	4	4	4	4	5
5. Loss of dissolved value per ton of ore	$ .04	$.044	$.026	$.080	$.044	$.026
6. Percentage of dissolved value saved	98.9	98.9	99.3	99.0	99.4	99.6

Undoubtedly within a short time there will be data of actual operations by this method published to show how closely the theoretical results have been attained in practice.

JOHN V. N. DORR.

Denver, Colo.

Ferro Titanium.

To the Editor of Metallurgical and Chemical Engineering:

SIR.—In your issue of July, 1911, page 348, there is an article by Dr. Hans Goldschmidt, in which he speaks of alloys produced by his aluminothermic process, and contrasts them with other alloys.

Regarding Ferrotitanium, he says: "This carbon-free ferrotitanium, containing 25 per cent of titanium, is used now quite extensively as an addition to steel for making rails and rolls. This ferrotitanium made by the aluminothermic process dissolves very easily and uniformly in the molten steel. It has a small intentional content of aluminium as this increases its solubility.

"The experience of metallurgists has shown that as a rule carbon-free metals dissolve more easily than metals containing carbon, since the latter contain carbides—that is chemical compounds with carbons."

It has been my privilege to have considerable experience with ferrotitanium alloys, in both steel and iron, which has been chiefly gained with the so-called 10 per cent and 15 per cent alloys of the Titanium Alloys Manufacturing Company. Based on this experience, these alloys can be thoroughly recommended. As is well known, these domestic alloys are not carbon-free, as they contain from 6.00 to 7.50 per cent total carbon, and so would appear to come under Dr. Goldschmidt's classification of alloys containing carbides. This, however, is not the case, for almost the whole of the carbon is present in the free or graphitic condition. The carbon in chemical combination varies from not more than 0.1 to 0.25 per cent, so that the amount of titanium carbide present, if any, must, therefore, be very small. The action of titanium, is some respects seems to be a good deal like that of silicon decomposing the carbide of iron, and precipitating the carbon as graphite.

When the 10 per cent or 15 per cent alloy is added to a bath of molten steel solution quickly takes place. Part of the graphite probably unites with iron forming carbide of iron, while a part rises quickly to the surface because of its low specific gravity, and is lost in the slag.

This opinion is supported by the results obtained in practice, for the carbon in the final steel when these alloys are used is generally slightly raised, but not to the amount expected if all the graphite were absorbed. It is extremely improbable that at the temperatures worked with, any titanium carbide is formed.

The Goldschmidt alloy when analysed shows that a considerable amount of aluminium is present and most probably unavoidably. This is present only in traces in the Niagara Falls alloys. In one sense its presence may be beneficial, but in another it is harmful. It may help the solubility of the low-carbon when added to the molten bath, and in this respect its action is necessary and helpful. After fusion, most of the aluminium will be oxidized to dry powdery alumina. This will tend to slag off, together with the oxide of titanium and other slag-making materials. Its action will be to decrease the fusibility of the slag, making it liable to be held in the metal, and so tend to defeat one of the chief advantages of using ferrotitanium, namely, a final clean solid steel or iron.

G. B. WATERHOUSE.

Lackawanna Steel Co., Lackawanna, N. Y.

Expropriation of Mining Rights.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I invite the friendly criticism of your journal on my proposed new (old) legislation permitting expropriation of mining rights (even *in invitum*) by strangers to the title in the soil. Some such legislation seems demanded and essential to the proper development of the mineral resources of States like those of the Atlantic seaboard, where obtains the ancient English common-law theory that ownership of the soil carries with it the title to the centre of the earth.

Rockland, Maine. C. VEY HOLMAN.

[As noticed in our May issue, page 278, a paper was presented by Mr. Holman at the last meeting of the American Electrochemical Society on the advantages of Maine for electrochemical industries, giving details of the available water powers and mineral resources of Maine. This paper was pub-

lished in full in Vol. 19 of the Transactions of the Society, page 351. Mr. Holman's argument is that the development and exploitation of these mineral resources seems wholly impossible under the present laws, owing to the unprogressiveness or unwillingness of their nominal owners, the holders of the land below which they lie, to undertake their proper working. For this reason Mr. Holman intends to have a bill introduced at the next session of the Legislature of Maine, providing a method by which mining of ore bodies in the lands of owners who are unable or unwilling themselves to develop them can be legally undertaken by strangers to the title who are willing to assume the risks. It is interesting to note in this connection that in his address on the mining industry and the public lands, delivered on Aug. 9 at the Michigan College of Mines, the Director of the U. S. Geological Survey, Dr. George Otis Smith, recommended that there should be made possible by legislation "the separation of surface and mineral rights wherever the two estates have values that can be separately utilized."—Editor.]

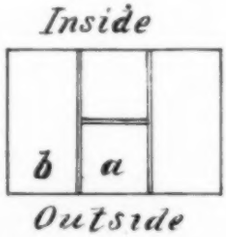
Flow of Heat Through Furnace Walls.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the recent Bulletin No. 8, by Messrs. Ray and Kreisinger, issued by the Bureau of Mines, and noticed briefly in your July issue, page 382, the conclusion is drawn from their experiments that a solid wall is a better heat insulator than one of the same total thickness having an air space not filled with any material, especially when this air space is near the hot side; also that when such an air space is filled with a loose granular material it insulates better than a solid wall.

These conclusions, if correct, seem to be quite important and valuable, although the most important one, namely, the one referring to the better insulation of loose, granulated material, is not new.

It seems to the writer, however, that the subject needs further investigation, as there are some discrepancies which seem to require an explanation. It is a well known fact, known probably to all blacksmiths, that when an iron bar which is cracked transversely is heated on one side of the crack, it may become red hot up to the crack, while it is still black on the other side. The crack may be so small as to be invisible to the eye, in the cold piece; in fact, it is believed that this phenomenon has been used to indicate the existence of invisible cracks.



This means, of course, that there is a very pronounced drop in the heat gradient at the crack, and this in turn means that the resistance to the flow of heat across the air space of the crack (or the counterpart of resistance when the heat is transmitted by radiation) is enormously greater than that of the iron. This is apparently in direct contradiction to the conclusions drawn in that bulletin. An air space is a wide crack.

A possible explanation of the discrepancy may be that the iron has such a very good thermal conductivity that the air space is poor relatively to it, and that the conductivity of fire brick is so very much poorer than iron that it is better than air. This, however, seems hardly likely, for it is evidently the air spaces in the interstices of the loose granular material that gives it its higher resistance. Moreover, the writer has an indistinct recollection having noticed that a glass rod with a crack behaved like the iron one, and the thermal resistivity of glass is of the same general order as that of refractory materials.

Moreover, the writer noticed recently in an electric furnace that the temperature of the brick at *a* in the above sketch, Fig. 1, was considerably cooler than that at *b;* the difference was so great that it was easily noticeable to the touch. This was no doubt due to the joint which separated the one from the inside of the furnace. If such a cheap and simple thing as a joint is such a good heat insulator, and this appears to be so, it would be unfortunate if the conclusions drawn in that bulletin were to be interpreted to the contrary.

It might be interesting to try heating the end of a transversely cracked fire brick or rod of insulating material to see whether it would behave like the iron bar. It would probably have to be done in the dark; moreover, the cooler end ought to be water cooled and the hot end ought to be heated quickly, for in course of time the heat would no doubt find its way to the colder side by passing around the end of the crack. Mere differences of temperature alone are not measures of thermal resistances any more than voltages are measures of electrical resistances.

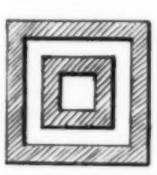


Fig. 2

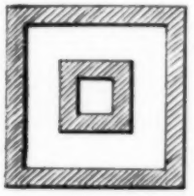


Fig. 3

If the electrical conductivity is any measure or indication of the thermal conductivity it may be of interest to note here that the writer has repeatedly drawn an alternating-current arc from the surface of a hot magnesite fire brick which formed a part of the furnace wall, the metallic charge in the furnace being the other pole of the circuit. The magnesite was scarcely visibly red.

According to the above-mentioned bulletin, filling an air space with loose granular fire brick gives that space an effective resistance which is greater than either of its two constituents, air and fire brick, would have by themselves. This seems to call for an explanation. Perhaps it lies in the resistances at the surfaces of contact of air and fire brick, as this surface is enormously greater for granular materials.

The following is another curious deduction from the formula in this bulletin which leads to apparently absurd conclusions, although they may be correct. Let Figs. 2 and 3 represent two double-walled furnaces, in the form of cubes, which are alike in every respect except that the air space within the walls in Fig. 3 is double that in Fig. 2, the thickness of each of the two parts of each of the brick walls being the same in both.

Let it be assumed for a first trial calculation that the leakage of heat is the same in both. The temperatures at the two bounding surfaces of the air space will then, of course, be the same, as the amount of heat radiated is independent of the thickness of the air space. But the cross section of the outer wall perpendicular to the flow of heat is much greater in Fig. 3 than in Fig. 2; suppose it to be twice as great in this particular case, then the drop of temperature in it would be only half as great as in Fig. 2 because the flow is the same, hence the temperature of the outside of the furnace would be considerably greater in Fig. 3 than in Fig. 2; the larger one would be the hotter. But with a higher temperature of the outside surface more heat would pass into the air; besides, the surface is now also larger, which still further increases the heat flow.

It follows, therefore, that the heat flow or leakage will not be the same in the two cases, as was assumed at the start, and that it will be *greater* for the furnace shown in Fig. 3. In other words, a larger air space insulates worse than a smaller one, which is apparently paradoxical. Yet it seems to lead to the conclusion that the thinner the air space the better the insulation; hence a joint between two abutting bricks ought to be a good insulator.

The above illustrations seem to show that it is not safe to jump to conclusions too quickly, as the problems may not be as simple as they appear at first sight.

Philadelphia, Pa. Carl Hering.

The Counter-Current of Continuous Agitation, Decantation, and Dilution Applied to the Cyanide Process

By JOHN E. ROTHWELL.

The present practice in the treatment of a gold and silver ore by the cyanide process, when using the agitation method, requires the mixture of a barren cyanide solution with the ore at its highest value; then by the continued agitation of this mixture, dissolution of the gold and silver is accomplished and the metal transferred to the solution, the rate of dissolution proceeding more slowly as the point of chemical equilibrium is approached. Thus it is necessary to agitate the mixture for such length of time as required to obtain the highest economical transfer of the metal into solution, and in some cases it means a retreatment of the charge after the removal of the first solution, in order to obtain a satisfactory recovery of the metal content of the ore.

The ideal and logical method of continuous treatment and operation is the reverse of this method—that is, the solvent lowest in metal value and highest in free cyanide should be applied to the ore after the greater value in metal has been dissolved. This solution is then separated from the ore and applied to ore of higher value, and so back up the line of treatment until the highest grade of solution is in contact with the ore at its maximum value, after which it is separated and passed to the precipitation department, where the metal content is precipitated and recovered and the barren solution returned to the mill system to go through the work of gathering up value again in the same sequence of operations.

It is known that if cyanide solution has approached the point of chemical equilibrium on any single quantity of ore, and is then withdrawn and another solution containing free cyanide, but not necessarily barren in metal value, is brought in contact with this ore, a further dissolution of the metal takes place that it was not possible to obtain with the first solution, and that in the repetition of this sequence of operations the point of saturation is approached.

Based on this condition, which in reality is simply the application of the "phase rule," I have designed a method for carrying out these operations in a mechanical and continuous manner.

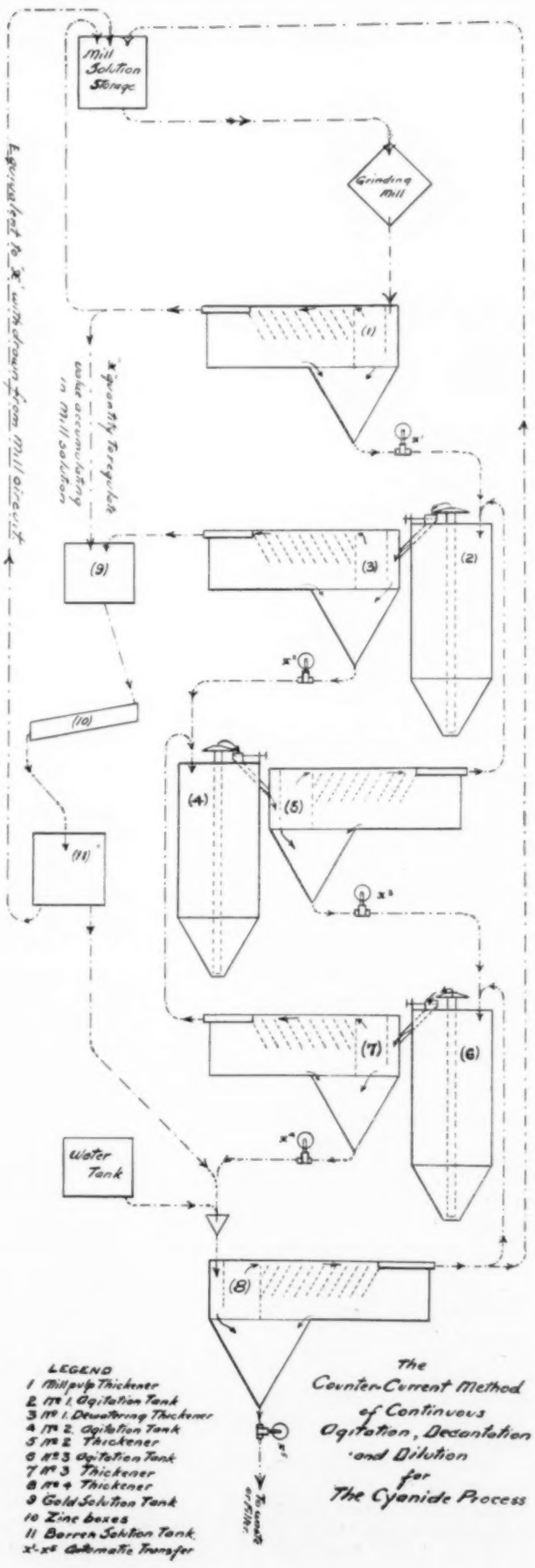
The method involves no change in the class of equipment employed in the ordinary continuous method of agitation, decantation and dilution as illustrated in a former article* of mine covering the continuous treatment of an ore, and as generally employed in mills using agitation tanks and dewatering thickeners. The only requirement is a re-arrangement of this equipment.

Reference is made to the graphic flow sheet accompanying this article, which illustrates the new arrangement, and the counter-current flow of solvent through the stream of ore. In this illustration, it will be noted that instead of placing the agitation tanks in continuous series, followed by the continuous decanting and diluting tanks, we place the agitation tank between the thickeners. The agitator then receives the thickened pulp from the thickener ahead of it, and the decanted solution from the thickener following it; thus it will be observed that the solvent is taking up value from the ore of lowest value and passing it up to the tank containing a higher value, while the ore is losing value as it passes down through each tank of the series toward the discharge.

The highest efficiency is obtained by having as thick a pulp as possible in the underflow, so that a minimum of dissolved value is contained in the moisture passing with the solids; and the greatest dilution is obtained from the counter-current flow, also the point of chemical equilibrium is removed as far as possible.

In describing the method and equipment I will not take up

*This journal, Vol. IX, No. 7, July, 1911.



the preliminary work of the preparation of the ore, as this will have to be suited to the requirements of the ore, with reference to amalgamation, concentration, fine grinding or preliminary treatment by oxidizing or other solutions to prepare the ore for the subsequent treatment in the cyanide solution.

The flow sheet shows a simple ore ground in a weak cyanide solution, and a comparatively small extraction of the metal values during the preparation.

For the operation of the counter-current method the agitation tank that can be used to the best advantage is the type in which the rising current of pulp is carried up through a central tube. At the top of this tube is a distributor, over which the flow of pulp passes, and a proportional part of this flow is cut out with a sampling vane located above the surface of the pulp in the tank. This position of the sampling vane forces the incoming flow of pulp and solution to pass from the top to the bottom of the tank before any part of it can possibly pass through the vane to the thickening tank, thus insuring a thorough mixture and agitation. The distributor may be of the rotating type or simply an inverted cone of definite diameter at the base. The sampling vane is located just below the perimeter of the cone, or in the concentric path of the rotating distributor, thus a uniform sample of the content of the tank is passed to the thickener. The adjustment feature of the vane permits of controlling the level of the pulp in the tank so the amount of outflow is equal to the inflow.

The thickener tank that works into the method is the one shown in the diagram, as it permits of a close and short connection to the agitator. This is a tank with a large cone and extended rectangular settling tank with inclined baffles that give it a very large settling area, in which the finest solids have a maximum time to settle and give a clear overflow at the weir. The weir is placed at the opposite end of the tank from the feed and the solids that settle to the flat bottom of the tank are moved to the cone by a slow-moving reciprocal rake. The large cone forms a receptacle for the storage of considerable thick pulp, thus insuring a very thick product, also a place in which any fluctuations in the flow from the grinding mill can be taken care of and the flow through the rest of the system maintained uniform.

The automatic transfer is a special type of diaphragm pump, capable of handling a very thick pulp, and may be regulated both in speed and stroke to give a definite quantity of feed in a unit of time.

In the arrangement of the series of thickeners and agitators to obtain the greatest economy in power, the tanks (2) and (3) of the series will be the lowest in the mill, the top of (4) and (5) will be about 2 ft. above the top of (2) and (3), while the top of (6) and (7) will be about 2 ft. above (4) and (5); (8) will be only high enough for the overflow to go by gravity into (6). The water tank and (11) will be located above (8); a meter placed in the outflow pipe from (11) and water tank regulates the flow of diluent into the series and registers the quantity passing. The automatic transfer takes advantage of the hydrostatic head in the thickener, thus reducing the lift necessary to discharge the pulp into the following agitator. The thickened pulp being the smallest quantity to be handled, means the minimum of power required in the system. If we figure, for example, a plant treating 100 tons a day, and the transferred pulp contains 50 per cent moisture, we then have approximately 3.26 cu. ft. flow per minute, equal to approximately 280 lb. to be raised an average height of about 6 ft. at each transfer, or 1680 ft.-lb. With the five transfers shown in the flow sheet this would be 8400 ft.-lb. Allowing 100 per cent for friction, we require very little over ½ hp for the whole transfer.

The arrangement also contemplates the overflow from (3) by gravity into (9), thence to (10) if zinc boxes are used; the barren solution is then pumped to (11). The elevation of this tank is only enough to flow back to (8) and need not exceed 30 ft. If we follow the example of 100 tons being treated, and take the quantity of solution precipitated at 3 tons for one of ore, *i. e.*, 300 tons per day, this will require, if a pump is operated continuously, 12,500 ft.-lb. per minute, which, allowing 100 per cent for friction, is about 2/3 hp.

If we take the power required to operate the air compressor for agitation in the agitation tanks at 5 hp per tank, this being about an average in the usual practice at the present time, and using three agitation tanks, we have 15 hp for this purpose. Allowing liberally for friction and contingencies, 20 hp will be ample for the whole work, or about 1/5 hp per day-ton for this department of the mill. Present practice is considerably in excess of this figure.

For the flow of the ore and counter-current flow of the solvent, reference is made to the diagram.

The following descriptive flow sheet will give the method of operation:

(1) Mill pulp thickener; receives all the flow from slime classifiers, overflow goes back to mill solution tank, except the small amount sent to (9) for precipitation if there is a tendency on the part of this mill solution to build up in value. This solution in some cases instead of being sent to (9) might be sent to (7) or (5), whichever tank should happen to contain a nearly equal amount of value per ton of solution. The underflow will be transferred in definite quantity and specific gravity to (2).

(2) No. 1 agitation tank; receives thickened pulp from (1) and counter-current solution from overflow of (5), discharges content continuously through sampling vane into (3).

(3) No. 1 dewatering thickener; receives flow through sampling vane from (2), clear overflow goes to (9), thick underflow by automatic transfer to (4).

(4) No. 2 agitation tank; receives thick plup from (3) and counter-current solution from overflow of (7), discharges content in equal quantity as received, through sampling vane, into (5).

(5) No. 2 dewatering thickener; receives flow continuously through sampling vane from (4), clear overflow goes to (2), while thickened underflow goes through automatic transfer in definite quantity and specific gravity to (6).

(6) No. 3 agitation tank; receives thick pulp from (5) and counter-current solution continuously from overflow of (8), discharges content in equal quantity as received, through sampling vane, to (7).

(7) No. 3 dewatering thickener; receives flow continuously through sampling vane from (6), clear overflow goes to (4), while thickened underflow goes through automatic tranfer in definite quantity and specific gravity to (8).

(8) No. 4 dewatering thickener; receives thickened pulp underflow from (7), mixed with barren solution from (11), plus water to make up moisture passing into the system from (1); overflow to the amount to maintain quantity in counter-current goes to (6), balance of overflow goes to mill solution storage; underflow goes by automatic transfer to waste, if sufficiently reduced in value, otherwise to an additional thickener, and this thickener would receive the flow from (11) and the water tank, instead of (8). The overflow would then go in part to (8) and the balance to the mill solution tank, while the overflow from (8) would all go to (6).

(9) Gold solution storage tank; receives overflow from (3), and delivers as required continuously to (10).

(10) Zinc boxes or zinc dust precipitating equipment; receives solution from (9), delivers barren solution to (11).

(11) Barren solution storage tank; receives solution from (10), discharges through regulating meter to (8) or subsequent thickener.

If the thickened pulp from the last dewatering thickener carries an amount of cyanide solution that would involve an excessive loss, this pulp can be washed on a pan or rotary type of vacuum filter and the cyanide solution recovered and returned to its proper place in the circuit.

The excess of cyanide can also be recovered to a high per-

centage by using two or three additional thickeners, and giving a dilution wash with water, and counter-current the wash against the solids, the overflow from the first tank going to the mill circuit to replace the amount drawn from it into the agitation circuit. A close regulation will obviate any accumulation of solution, particularly if the ore treated settles naturally to a fairly dry pulp.

There are many advantages apparent in this system as a whole over the present method of operating, the important ones of which are as follows:

The dissolved values are recovered from the system at the point farthest removed from the discharge of the tailings; thus it is impossible for a charge of pulp containing high value in solution to be thrown away by an imperfect wash or carelessness or neglect on the part of the attendant operating a filter as in the present method.

The amount of solution carried in the circuit is less than in the present method, while at the same time any unit of ore is in contact with a larger volume of solution than at present.

By proper regulation, a minimum of solution of constant maximum value is sent to the precipitation department, so that there is no occasion to carry two or more circuits for precipitation purposes.

There will be no accumulation of solutions, from which values have to be precipitated and the solution thrown away.

The power required for operating will be less and the equipment less for given tonnage capacity, thus the first cost will be less for installation per day-ton treated.

The counter-current system can be controlled in all its operation from a central point, which registers and records the exact flow at all times and sounds an alarm if any stoppage occurs or a danger point is reached that would throw the regular operation out of equilibrium.

A complete control of the process is possible by taking samples of the pulp and solutions passing, and making the necessary determinations of value considerably in advance of the time of discharge of the pulp as waste. As the mechanical operations are simple and automatic, the labor required around such a plant is a minimum, so that the personal equation plays a very small part in the whole.

The system may be applied to the treatment of ores other than gold and silver where a hydro-metallurgical method can be applied.

Denver, Colo.

New Method for the Rapid Determination of the Total Carbon in Iron, Steel and Ferro-Alloys.

BY A. L. QUENEAU.

It is of great importance to be able to obtain rapid as well as accurate total carbon determinations in operating steel open-hearth or electric furnaces. The colorimetric method of Eggertz was formerly the quickest known, though it is not as reliable as the usual combustion method in a porcelain tube in a stream of oxygen.

The method herein described is due to M. De Noully, chief chemist of the St. Chamond Steel Works, in France. It permits of the same accuracy as the standard combustion method while giving the results in about 5 to 7 minutes. Furthermore, the apparatus used is extremely simple and can be installed independently of a regular laboratory in close proximity to the steel furnaces.

The method is applicable to the various grades of pig iron, steels and ferro-alloys; the accuracy of the results being entirely independent of the presence of nickel, chrome, tungsten, molybdenum, vanadium, titanium, sulphur or phosphorus.

Principle of the Method.—The prepared and weighed sample of the steel or ferro is burned in an atmosphere of oxygen, the combustion being started by means of a small arc flashed through the steel sample. The resulting CO_2 is absorbed in a known volume of standard NaOH solution, the excess of free NaOH being titrated by means of a standard sulphuric acid solution; phenolphthalein is used as indicator.

Solutions Used.—Alkali Solution.

NaOH	4 grams
H_2O	1000 cc.

Indicator.

Phenolphthalein	1 gram
Alcohol, 90°	100 cc.

Sulphuric Acid Solution.

H_2SO_4	4.083 grams
H_2O	to make 1000 cc. of solution

The acid solution contains 3.333 grams of SO_2 per liter; 1 cc. therefore corresponds to 1 mg. of carbon. As usual, the acid solution is standardized with a $BaCl_2$ solution.

Apparatus.—The apparatus consists of:

1. A liter Erlenmeyer flask of Jena glass, closed by a rubber stopper with 3 holes, through which pass 2 copper electrodes

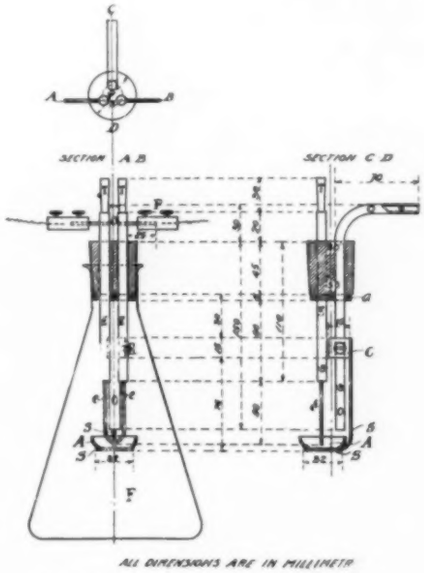


FIG. 1.—DIMENSIONS OF APPARATUS

F—Erlenmayer Flask of 1 Liter.
B—Rubber stopper with 3 holes.
a—Asbestos disc to protect the rubber stopper from any projections during combustion.
E—Electrode supports.
e—Electrodes.
T—Fiber handles allowing to vary at will the electrode gap.
P—Current terminals.
O—Copper tube for the oxygen.
C—Adjusting sleeve with set screw.
S—Support for the asbestos boat.
A—Asbestos boat.

and a copper tube. To the copper tube is attached a copper wire, forming a loop to support an asbestos boat and a basket of copper wire cloth placed under the asbestos boat. The basket is to receive any material that may escape in case of the asbestos boat being bored by a too violent combustion of the metal sample. The copper tube terminates a short distance above the boat and serves to bring into the flask a supply of oxygen and directly over the material under analysis.

2. A glass bottle reservoir for the oxygen, which is maintained at a slight pressure, 30 to 40 cm. of water. The water is rendered slightly alkaline with KOH in order to free the oxygen of whatever traces of CO_2 may be present in the gas as supplied.

3. A small glass bottle forming a mercury trap to prevent any gas passing back from the Erlenmayer flask in case of too intense combustion.

The asbestos boats are readily made from asbestos paper 1 or 2 mm. in thickness, from which discs, 50 mm. in diameter, are cut. These discs, slightly moistened, are pressed by means

of a bronze former, 25 mm. outside diameter, in a steel die, 30 mm. inside diameter. The boats are thoroughly dried and are also calcined in a muffle furnace to decompose any carbonates that may be present in the asbestos.

Preparation of the Sample.—Whenever possible the samples are obtained by drilling at high speed with a suitable twist drill; thin voluminous drillings are desired; dust and needle-like drillings are to be avoided if possible.

1. The hard or tempered steels are crushed and passed through 100 mesh.

2. All grades of pig iron are crushed and passed through 80 mesh.

3. All ferro-alloys are crushed and passed through 120 mesh.

For soft steels a good combustion is readily obtained without the addition of any oxidizing agents; for all other metals it is necessary to add a certain amount of PbO_2 or Bi_2O_4 to the samples, as will be shown in the table given below.

With all materials containing over 1.5 per cent of carbon in order to secure a complete combustion it is necessary to add to the sample a certain amount of fine drillings from an extra

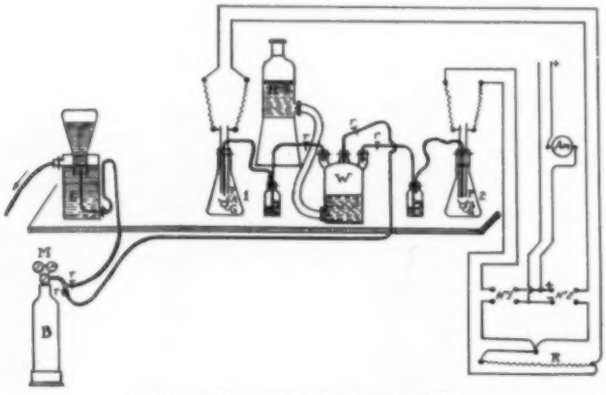


FIG. 2.—ARRANGEMENT OF APPARATUS.

A—Asbestos boat.
G—Copper wire cloth basket or clay dish to collect any oxides that may escape in case of a rupture of the asbestos boat.
F_1 & F_2—1 Liter Erlenmayer flasks.
B—Oxygen tank.
M—Reducing manometer.
C—Water tank used in filling the flasks with oxygen.
W—Wolf bottle used as oxygen reservoir.
H_2O—Water bottle to give the necessary pressure to the oxygen in W.
Hg—Mercury traps.
Am—Ammeter.
r—Cocks.
R—Electric resistance.
No. 1 & No. 2—Two-throw switch with fuses.

soft and pure steel (.05 to .07 C.), due corrections being made in the results obtained for the additional carbon introduced, the determination of the carbon contents of the extra soft steel employed having been made once for all.

Materials.	Wh. of S'ple gram.	Wt. of PbO_2 gram.	W'ght of Soft Stl. gram.
Steels with less than 1.5% C....	1 or 2.	.100 or .200[1]	
Hard steels or steels with over 1.5% C.	1.	.200	
Pig iron	.500	.200	1
Ferro-chromes with 6 to 10% C.	.250	.200	2
Ferro-chromes with 1 to 5% C..	.500	.200	1
Ferromanganese	.250 or .500	.200	1 2
Ferrosilicons, 10 to 90%.....	.500	.200	1
Ferro-molybdenum	.500	.500[2]	1
Ferro-tungsten	.500	.100[3]	1

[1]With soft steels it is not necessary to add any PbO_2 unless the borings are coarse and irregular in size.
[2]With less PbO_2 some molybdic acid is volatilized and is absorbed by the NaOH solution.
[3]With more than .100 of PbO_2 the combustion is too intense and the asbestos boat becomes bored thereby.

If any extra soft steel borings are required these drillings are spread on the bottom of the boat and on top of the layer thus formed is placed the mixture of the pig iron or ferro with the lead peroxide.

The above table will enable one to obtain the complete combustion of all products, even gray iron containing a large amount of graphite.

For the analysis of all products with more than 1.5 per cent of C it is recommended that the NaOH solution be diluted to 500 cc. in order to make the end reaction more sensitive and also to bring back the pink color of discoloration by the addition of a known quantity of the NaOH solution. With high carbon contents it is also well to take 200 cc. of the NaOH solution, instead of 50 cc., in order that there may be a large excess of free alkali to absorb the last traces of the formed CO_2.

The results are not affected in the least by the presence of sulphur. A test was made with samples containing .80 per cent of S. In the same manner the presence of phosphorus is not detrimental; phosphoretic pig with 2 per cent P gave accurate results with this method. The sulphates and phosphates formed remain *in toto* in the asbestos boat, from which they may be collected and then determinations made.

Manipulations.—The asbestos boat containing the sample to be analyzed is placed upon its support, the height of the boat being so adjusted that its edge level is from 5 to 10 mm. from the level of the end of the supporting copper tube. The copper electrodes are placed with the free extremities buried in the sample and about half way therein. The rubber stopper with its adjuncts is placed in the neck of the Erlenmayer flask, a careful closure of it being made without disturbing the boat or its contents. The flask has been previously filled with oxygen gas and has also received 40 or 50 cc. of the NaOH solution. The various connections are now made and the cock *r* is opened, thus establishing communication with the oxygen reserveir *W*; the electric current is then made to pass across the gap of the electrodes by closing the double-throw switch. The resistance *R* is such as to permit, with a given source of electric current, the passage of 12 or 15 amp when the electrodes are in short-circuit.

As soon as the combustion has begun the current is cut off, the metal continuing to burn until complete combustion has taken place.

In the event of the combustion not starting properly the gap between the electrode points is changed by means of the fiber handles *T*, while the current is sent in repeatedly by operating the switch.

If the combustion is too rapid the cock *r* is closed, cutting off the access of the oxygen.

The combustion being complete, the cock *r* is closed, the flask is disconnected and it is cooled by plunging it in water; when cooled it is rapidly opened, the stopper with the boat removed and a plain stopper is substituted. The flask and its contents are thoroughly agitated for 10 seconds and the free NaOH is titrated by means of the sulphuric acid solution. As a check on the alkali contents of the NaOH solution a fresh sample is now titrated and the difference of the two titrations, the same volumes of the alkali solution having been taken, corresponds to the NaOH absorbed by the CO_2 formed. If the acid solution contains exactly 3333 grams of SO_3 per liter, 1 cc. of it corresponds to 1 mg. of C.

By taking the difference of the two NaOH titrations it is seen that no correction need be made for any carbonation which is bound to occur in the NaOH solution.*

The Arizona Copper Company, Ltd., produced during the half year ending March 31, 1911, 9,133 tons of Bessemer copper. The quantity of copper bearing material, ore and concentrate, smelted during the same period was 80,643 tons. The copper yield was 11.32 per cent on the gross tonnage treated.

*Bulletin de la Société de l'Industrie Minerale, November, 1910; July, 1911.

Refining of Silver and Gold Alloys

By EDWARD F. KERN, PH.D.,
From Lecture Notes Prepared for the School of Mines of Columbia University.

One of the final products which is obtained by the treatment of anode sludge by the cupellation of retort-bullion from the Parkes desilverization process, by the cupellation of the final "mother-metal" of the Pattinson process, and by the treatment of silver-gold ores, is an alloy composed chiefly of silver and gold, and associated impurities such as copper, selenium, tellurium, and traces of lead, bismuth, etc.

If the alloy contains an appreciable amount of one or more of the following metals: lead, bismuth, antimony, arsenic, selenium and tellurium, it should be purified as completely as possible, as these metals interfere with and increase the cost of separating and purifying the silver and gold. This is accomplished by either oxidation in the cupellation furnace, or else by melting the alloy with an oxidizing flux, such as potassium nitrate.

If the alloy contains over 75 per cent of silver, it is known as doré bullion. It is refined and the gold recovered by either the acid-parting process or the electrolytic-parting processes.

If the gold content of the alloy is over 25 per cent, the direct treatment by either of these processes cannot be economically accomplished. When this is the case, the alloy is melted with a sufficient amount of impure silver (or doré bullion) so as to dilute the gold content of the resulting alloy below 25 per cent.

In order to accomplish rapid parting of the silver from the gold, the doré bullion is cast into thin plates of suitable size and shape for treatment. The thinner the plates, the more rapid the parting is accomplished.

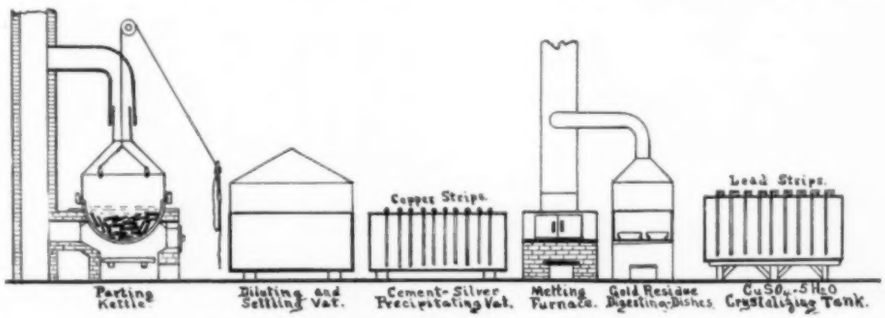


FIG. 1.—SECTIONAL SKETCH OF ACID PARTING PLANT.

Parting of Doré Bullion with Boiling Concentrated Sulphuric Acid.

The doré bullion, as thin plates containing 75 to 98 per cent silver, 25 to 1 per cent gold, and 1 to 2 per cent copper, selenium and tellurium, is placed in cast steel or cast-iron kettles and digested with boiling concentrates H_2SO_4 (66° B) for 15 to 20 hours, until only fine particles are left. (If the alloy contains over 40 per cent gold, it dissolves very slowly.) The reactions for dissolving are:

$$2\,Ag + 2\,H_2SO_4 = Ag_2SO_4 + SO_2 + 2\,H_2O.$$
$$Cu + 2\,H_2SO_4 = CuSO_4 + SO_2 + 2\,H_2O.$$

The usual capacity of the kettles is from 10,000 to 20,000 Troy oz. of doré bullion per 24 hours; 2.5 lb. of concentrated H_2SO_4 are required per 12 oz. of bullion.

As $CuSO_4$ is only slightly soluble in concentrated sulphuric acid, the copper in the doré bullion must be less than 8 per cent, otherwise crystals of $CuSO_4$ will coat the metal and prevent it dissolving. Ag_2SO_4 is, however, soluble in one-quarter its weight of boiling concentrated H_2SO_4; whereas, in dilute (5 per cent) H_2SO_4 at 90° C. it is soluble in 25 parts, and in acid of the same dilution at 20° C. one part of Ag_2SO_4 is soluble in 100 parts.

During the dissolving of the silver, a large amount of SO_2 is formed; the kettles are, therefore, covered with an iron hood which is connected with a ventilating chimney. The heating of the kettles is done by means of a fire beneath.

When the silver is all dissolved, the fire is withdrawn from under the kettle, and a small amount of cold dilute sulphuric acid (specific gravity 1.6) is added to the hot solution. The addition of the cold dilute acid causes the precipitation of some Ag_2SO_4, which carries with it any suspended particles of gold. If much copper is present, the addition of cold dilute acid is not necessary, as some $CuSO_4$ will precipitate when the solution cools, collecting the suspended particles of gold in the same manner as the Ag_2SO_4. In some plants the hot solution is ladled into a second, heated, kettle, in which the suspended finer residue is allowed to settle. This procedure gives a more rapid recovery of the major portion of the gold, as the coarser residue which remained in the digesting kettle may be under final treatment during the time that the finer particles are settling from the solution which is in the settling kettle.

The hot solution, as soon as clear, is siphoned into about *20 times its volume of hot water* in a covered lead-lined vat. Any fine gold, which may have been carried over, is allowed to settle and the clear dilute hot solution is run into another lead-lined vat in which are suspended strips of copper, which precipitate the silver as "cement silver," $Ag_2SO_4 + Cu = CuSO_4 + 2Ag$. The solution is kept boiling by steam. The end of precipitation of the silver is determined by testing with HCl; if silver is in solution, AgCl forms.

The cement silver is washed free of copper sulphate, dried and melted. If any selenium or tellurium are present, some KNO_3 is added to oxidize and slag them. The refined silver is cast into bars weighing 1000 to 1200 Troy oz., which assay 998 + fine. The slag is added to charge in the smelting furnace, or placed on a charge of metal in the cupel furnace for the recovery of its silver content.

The $CuSO_4$ solutions are evaporated to crystallization for $CuSO_4.5H_2O$, and the mother liquors are re-evaporated to crystallization several times; the crystals are refined by dissolving in water and recrystallizing. The final mother liquors are evaporated to 55° B in lead pans, then transferred to an iron kettle and evaporated to 66° B. The concentrated acid is allowed to cool and the residue to settle; the acid is then siphoned off and used for parting doré bullion. The residue is collected and sent to the smelting furnace, or else put in the cupel furnace.

The *residue of gold* from the parting of the doré still contains some silver. It is transferred to a small iron kettle and again digested with boiling concentrated H_2SO_4 for four to six hours. The solution is poured off and added to the large parting kettle. The residue of gold is washed with hot water, transferred to a porcelain dish and digested with boiling dilute HNO_3 in order to remove the last traces of silver and copper. The solution is poured off and the silver precipitated on metallic copper. The residue of gold is well washed with hot water, dried and melted with a flux of borax-glass in a graphite crucible, and cast into bars (998 + fine). The slag is smelted, or else placed in the cupel furnace.

Some metallurgists dispense with the last HNO_3 digestion, and others digest first with boiling dilute HNO_3, after which the residue is finally digested with boiling sulphuric acid.

Electrolytic Parting of Doré Bullion.

The parting of doré bullion by sulphuric acid is rapidly giving place to the cleaner and less expensive electrolytic separation.

The parting of doré bullion containing 85 to 95 per cent of silver, 1 to 8 per cent gold and 2 to 6 per cent copper, is accomplished by making the metal an anode in an electrolyte containing from 2 to 4 per cent $AgNO_3$, 0.1 per cent to 1.0 per cent free nitric acid, and 1 to 4 per cent of $Cu(NO_3)_2$. The silver is dissolved, leaving the gold (Pt, Pd, Ir, Pb) intermixed with a small amount of finely divided silver as the anode sludge; copper, if present in the anode, dissolves along with the silver; it is not precipitated on the cathode, but accumulates in the electrolyte.

The anodes must be, as far as possible, ridded of lead, bismuth and antimony by cupellation, before the doré is cast into anodes.

The electrolytic parting of doré bullion is carried on either in the Moebius cell or in the Balbach-Thum cell, which differ in their construction. However the electrolytic principles in each are the same. The Moebius cell is the older form of apparatus.

The **Moebius cell** (Fig. 2) is a square wooden cell heavily coated with asphaltum, or an acid-proof earthenware cell (size about 24 in. by 26 in. and 22 in. deep.) In each cell are suspended four sets of four anodes (14 in. by 5½ in. by ⅜ in.) and five rolled sheet cathodes (cathode surface 18 sq. ft.).

Each set of four anodes is surrounded by a muslin bag, the purpose of which is to prevent the anode sludge becoming mixed with the deposited silver. Between each set of anodes and cathodes are two wooden stirrers, which operate back and forth for the double purpose of agitating the electrolyte, and to prevent the anode and cathodes becoming short-circuited by the long crystals of cathode silver. The stirrers knock the crystals from the cathodes, and they drop to the bottom of the cell, collecting in a wooden tray, which has a hinged bottom. The conductor bars of the anodes and the cathodes, the stirrers and the tray are all attached to a wooden frame, which may be raised from and lowered into the cell by means of a small windlass. Once or twice a day the current is shunted, the frame lifted from the cell and the silver scraped from the cathode and caught in the tray beneath. The silver is dumped from the tray, the frame then lowered into the cell and the current turned on. The refined silver is carried to the washing vat. The dried silver crystals are melted in a graphite crucible or in an oil-fired reverberatory furnace and cast into bars, assaying 999 + fine silver.

The gold sludge in the muslin bags, which contains all the gold (Pt, Ir, Pd), most of the bismuth (as $Bi_2O_3xH_2O$), and lead (as PbO_2) of the original doré bullion, and a small amount of the silver and copper, is removed every second or third day. It is refined by acid treatment, or by the Wohlwill gold refining process.*

In the course of the electrolysis of doré bullion, its copper content dissolves along with the silver and accumulates in the electrolyte, replacing an equivalent of silver (here Cu stands above Ag in the e.m.f. series). The copper in the electrolyte may amount to as much as 5 per cent, and the silver reduced to as low as 0.5 per cent without the copper being precipitated along with the silver. When the copper reaches about 4 per cent, part of the electrolyte is withdrawn from the cell, and it is replaced by a fresh lot of $AgNO_3$ solution from the treatment of the anode sludge.

*See papers by E. Wohlwill, this journal, vol. II, pp. 221 and 261; vol. VI, p. 450, and by Tuttle, vol. I, p. 157; also vol. VI, pp. 355 and 408 and vol. VIII, p. 82.

In some cases, when the copper content of the anode is low, the electrolyte which adheres to the cathode silver is ample to keep the copper in the electrolyte sufficiently low to prevent it depositing with the silver. The silver in the withdrawn solution is precipitated on sheets, or plates, of copper, giving cement silver, which is collected and melted with the washed cathode silver. The copper in the precipitated solution is recovered by either precipitating on metallic iron, or else the solution is thrown on roasted ore or matte which is to be smelted.

The Moebius cells are usually connected in multiples of five or six. The current per cell is from 350 to 450 amp. (C. D. = 20 to 25 amp. per sq. ft.) and e.m.f. per cell is 1.4 to 1.7 volt. The current efficiency is from 92 per cent to 95 per cent. The amount of silver permanently tied up as cathodes and in electrolyte is from 40 to 45 per cent of daily capacity; but by operating with higher current density, the silver permanently tied up per daily capacity of the cell is reduced. A current density of 40 to 50 amp. per square foot has been successfully employed.

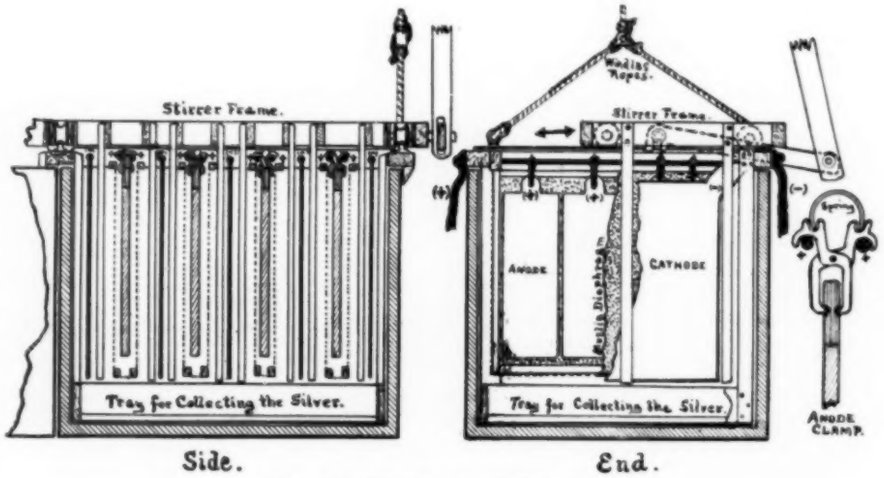


FIG. 2.—SECTIONS OF MOEBIUS CELL.

The power consumption in the Moebius cell is about 13.5 watt-hours per Troy ounce of silver deposited. Cost of refining in this cell is from ¼ to ½ cent per Troy ounce of silver deposited. The parting is rapid, as the anodes are dissolved on all sides; and when operating with high current density, the recovery of the refined silver is as rapid as by the sulphuric-acid parting process.

Dr. D. K. Tuttle (this journal, vol. IV, p. 306) and assistants, at the Philadelphia Mint, developed a method of electrolytically refining silver-gold alloys which gives solid, adherent, crystalline deposits of silver on the cathode, and thus obviates the need of mechanical stirrers, as are used in the Moebius cell. The process is in use at the U. S. Mints in Philadelphia, Denver and San Francisco.

The general arrangement of the cells and the operation of refining are similar to the refining of copper and of lead, but on a smaller scale. The cells are made of acid-proof earthenware, 20 in. wide and 11 in. deep, and hold 42 anodes (6 rows of 7 anodes side by side) and 40 cathodes (5 rows of 8 side by side), suspended vertically with 1.5-in. space between centers of electrodes. The anodes are 7.25 in. long, 2.5 in. wide and ⅜ in. thick. The cathodes are of same width and length as the anodes and are made of rolled sheet silver 0.016 in. thick. The electrodes in each cell are in multiple.

The cells are connected in series, and are operated at a current density of about 7 amp. per square foot of anode surface, and an e.m.f. of from 1 to 1.4 volt, with a current efficiency above 90 per cent.

The gold which is present in the anode in amounts between

15 and 30 per cent, remains attached to the anode as a chocolate colored mud. If the gold is present in smaller amounts, the anodes must be enclosed in muslin bags for the purpose of preventing the anode sludge floating in the electrolyte and becoming attached to the cathode.

The electrolyte contains about 4 per cent $AgNO_3$ and 1.0 per cent HNO_3. It is circulated by means of small propellers at one end of the cell, or by gravity. (The gravity cascade system of circulation gives better circulation than the propellers, and does not keep particles of the sludge in suspension so long a time as the propellers, therefore affording a higher recovery of the gold residue and the production of purer silver.)

By maintaining rapid circulation and by having present in the electrolyte 1 *part by weight of gelatine in* 8000 *to* 10,000 *parts of solution*, the silver deposits on the cathodes as a mass of solid coherent crystals, which seldom short-circuit the cell. The successful operation of the process depends upon the crystals remaining attached to the cathode and to the overcoming of short-circuits by producing smooth deposits. The addition of the gelatine is made each day as needed, which is judged by the appearance of the deposits. It is added as a solution in hot water (10 to 20 grams per litre). Too much gelatine darkens the electrolyte (due to the chemical precipitation of cement silver), and also causes the deposit to form non-coherent.

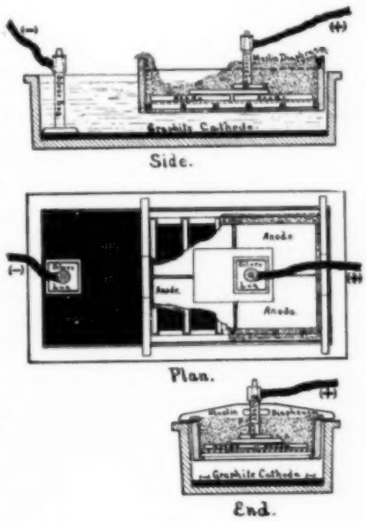


FIG. 3.—BALBACH-THUM CELL.

The cathodes, when the anodes are almost completely dissolved, are removed from the cell, washed, melted and cast into bars of 999 + fine. The gold residue is collected, washed, and treated by either the acid leaching method (when only silver is present), or else is melted, cast into anodes, and refined by the Wohlwill electrolytic process (in case Pt, Pd, and Ir are present).

The scraps of anode are remelted along with silver to be refined.

On account of the low current density which is necessary for the production of smooth, coherent deposits of silver, this process of refining cannot be economically used in the works. But as the alloys which are refined in the Mints contain from 15 to 30 per cent of gold, and as no interest is charged against the metal which is "tied up" in the cells, this method of parting silver and gold can be used in this special case.

The **Balbach-Thum cell** has no mechanically moving parts. It consists of a long, shallow, narrow cell about 10 in. deep by 26 in. wide by 48 in. long, made of chemical-proof earthenware. They are supported on acid-proof insulated pillars.

The cathode is made up of strips of Acheson graphite, about ¾ in. thick and completely covering the bottom of the cell. Surface, 8 sq. ft. A silver lug is used for electric connection.

The anodes of doré bullion, size 8 in. by 12 in., are suspended horizontally in the cell by being placed in a wooden or earthenware tray (24 in. by 24 to 36 in.), with muslin bottom, which is about 4 in. from the cathode. The muslin is supported on cross-bars which form the bottom of the trays. Electrical contact of the doré slabs is made by means of a slab of silver with a vertical lug.

The silver is deposited as loosely adhering crystals on the graphite slabs, and from time to time are removed with a hardwood shovel. The crystals are washed, melted and cast into bars assaying 999 + fine, and weighing 1000 to 1200 Troy oz.

The gold (and Pt), bismuth (as $Bi_2O_3.xH_2O$), lead (as PbO_2) collect as sludge on the muslin in the tray, and are removed every third day and treated in the same way as anode sludge from the Moebius cells.

The electrolyte is kept in proper condition by removing a portion, when copper reaches about 4 per cent, and replacing with same amount of fresh solution, in the same way as is done in Moebius cell.

The cells are arranged in sets of 4 to 6 in parallel, and operate with a current of about 150 amp. per cell.

The cathode surface is 8 sq. ft., giving a current density of 20 amp. per sq. ft.

The anode surface is about 4 sq. ft., giving a current density of 40 amp. per sq. ft.

The current efficiency averages 88 per cent; at times, 90 per cent.

The e.m.f. per cell ranges from 3.2 to 3.8 volts.

The power consumption in the Balbach-Thum cell is about 31.5 watt-hours per 1 Troy oz. of silver deposited.

The amount of silver permanently tied up in the cell is about 32 per cent of the daily output of the refinery.

Advantages claimed for the Balbach-Thum cell over the Moebius cell:

1. No moving parts, therefore less labor and apparatus to keep the plant operating.
2. Less silver tied up tn the plant (32 per cent as against 45 per cent).
3. Less time lost in removing the silver and sludge.
4. Less wear and tear on apparatus, therefore less repair costs.
5. No anode scrap to remelt. (The anodes allowed to completely dissolve in the cell).

Advantages claimed for the Earthenware Tray cells over the Wooden Tray cells:

1. Cleaner and more permanent cell.
2. Deposited silver forms as larger crystals which are easily washed and more readily melted. (The action of the electrolyte on the wood forms organic compounds which, when present in the electrolyte, causes the silver to deposit as fine crystals).
3. Higher current efficiency (93 to 96 per cent) as against 88 to 90 per cent).

Objections claimed for the Balbach-Thum cell as compared with the Moebius cell:

1. Larger power consumption (3.5 volts as compared with 1.5 volt).
2. Large consumption of HNO_3, due to high e.m.f., forming NH_3.
3. Larger space required per given daily capacity, (about 6 times more space).
4. The anodes dissolved only on under side, facing the cathode, therefore less rapid per anodes under treatment.
5. More silver tied up in the electrolyte per given parting capacity, (larger volume of electrolyte per weight of anodes).

Advantages of electrolytic parting over acid parting of doré Bullion:

1. Production of purer silver.
2. Complete separation of selenium and tellurium (these render silver brittle).
3. Less cost of operation, due to smaller consumption of chemicals.
4. Less loss of metal, due to less handling of materials.
5. $CuSO_4.5HO_2O$ not produced as by-product.

6. The purified silver recovered in shorter time, if high current density is used.

Disadvantages of electrolytic parting, as compared with acid parting of doré bullion:

1. First cost of plant (electric generators, conductor cells, etc).

2. Time the gold residue held for parting (interest on same). It requires from 48 to 60 hours to have the gold in bars, whereas about 30 hours required when the parting is done with sulphuric acid.

Treatment of Silver Anode Sludge.

There are two methods of treating the silver anode sludge:

A.—Chemical Method. When there is not enough Pt. Pd and Ir present to pay for recovery: The sludge consists of all the gold (Pt, Pd, Ir), most of the bismuth (as $Bi_2O_3xH_2O$) and lead (as PbO_2) contained in the doré bullion, and a small amount of silver and copper. It is treated in porcelain, or acid-proof, earthen-ware bowls, which are heated by steam jackets, which conform to the bottom of the bowl, or else are heated by being placed in a sand-bath with flame underneath. The bowls are covered with an acid-proof hood which is in connection with a good draft, for the purpose of carrying off the acid fumes.

The sludge is first boiled with dilute HNO_3 (2 acid: 3 water) till no further reaction occurs; the solution is decanted, the residue of gold washed, and retreated with boiling concentrator H_2SO_4. When acid treatment is complete, the solution is decanted and the residue thoroughly washed with hot water, dried, and melted under borax glass in a graphite crucible. It is then cast into bars assaying 998 + fine.

The HNO_3 solution, containing silver nitrate and free nitric acid, is diluted and added to the silver electrolyte in the parting cells in place of the solution which is removed from time to time in order to keep the copper content down.

The H_2SO_4 solution is diluted, and the silver precipitated by boiling with sheets of metallic copper, giving cement-silver, which is melted with the cathode silver and cast into bars. The resulting $CuSO_4$ solution is either crystallized, or else sent to the copper refinery and added to the copper electrolytic tanks.

B.—Electrolytic Method. When platinum, palladium, and iridium are in sufficient amounts to pay for their recovery: The sludge is washed free of electrolyte, dried, melted under borax glass and soda ash flux, and cast into anodes, which are refined by the Wohlwill electrolytic process.

Protection of Wood Work of the Parting Plant.

All exposed woodwork of the parting plant should be well protected with several coats of an acid-proof paint (asphaltum paint, or patented P. & B. paint) in order to prevent the possibility of fire by spontaneous combustion, due to the action of nitric acid fumes on the wood. Ample ventilation for the removal of the fumes should be provided also, in order to protect the health of the workmen.

Electrolytic Refining of Gold.

The Wohlwill Electrolytic Gold Refining Process is used at the U. S. mints in Philadelphia, Denver and San Francisco for purifying gold, and for the recovery of the valuable impurities (such as Pt., Pd., Ir., and Os).

The anodes are gold bullion, of reasonable purity, running over 940 fine, and the cathodes are thin sheets of hard rolled gold.

The electrolyte contains from 3 to 4 per cent $AuCl_3$, 2 to 5 per cent free HCl (sp. gr. 1.2) and 1 to 3 per cent NaCl. If the anodes contain lead, then from 1 to 2 per cent H_2SO_4 is added to prevent the lead dissolving and being deposited with the gold; the H_2SO_4 causes the lead to enter the anode sludge as $PbSO_4$.

The conditions necessary to obtain high current efficiency and to deposit the gold in solid form are the presence of free HCl in the electrolyte, circulation of electrolyte, and temperature between 50 and 70° C. With neutral $AuCl_3$ electrolyte and at ordinary temperature, the anode is slowly dissolved, free chlorine being liberated at the anode, and hydrogen at the cathode.

The circulation of the electrolyte is accomplished by means of small propellers in one end of the cell, or else by the gravity cascade system. The former method of circulation is not so satisfactory as the latter, as it keeps the anode sludge longer in suspension, which permits particles becoming attached to the cathode. In case of the gravity-cascade system of circulation, the cells are arranged in rows of 12 or more, with ample height between adjoining cells to allow the solution flowing from the upper cell to the lower one. Porcelain pumps are used for raising the electrolyte from the sump cell to the upper cell. The circulation should be ample to maintain uniform composition of the electrolyte in all parts of the cell, which may be judged by the corrosion of the anode and the appearance of the cathode.

The cells are acid-proof earthenware, or porcelain, about 15 in. by 12 in. and 10 in. deep (or larger, depending on size and number of anodes).

A sufficient current density is used so as to allow the refining being done as rapidly as by the acid refining process, which is in 24 hr. The usual current density is between 80 and 120 amp. per square foot. The e.m.f. per cell is between 1 and 1.3 volt, with electrodes spaced about ¾ in. centers. The energy consumed is about 10 watt-hours per 1 troy ounce of cathode gold.

The heating of the cells is accomplished by having them placed on a bed of sand, which has steam coils imbedded. At a temperature of 50 to 70° C. much evaporation occurs, so water must be added from time to time to keep the volume of electrolyte constant.

The anodes of impure gold (940 + fine) are cast in iron molds, size 6 in. long by 3 in. wide and about ⅜-in. thick (24 hour anodes). The cathodes are of same length and width as the anodes, and are made of pure rolled gold sheets.

The purity of the deposited gold is 999.8 fine. It is melted in graphite crucibles and cast into bars.

The sludge contains AgCl, $PbSO_4$, Ir. and Os. If the anodes run less than 940 fine (6 per cent Ag + Pb) they become coated with the AgCl and $PbSO_4$ which are non-conductors. In this case, it is necessary to scrape the anodes when voltage of cell becomes high and current becomes low.

The silver is recovered from the sludge by fusing it with Na_2CO_3, which reduces the AgCl to metal:

$$2AgCl + xNa_2CO_3 = 2Ag + 2NaCl + CO_2 + O + (x\text{-}1)\ Na_2CO_3$$

If iridium and osmium are present in the anode, they collect with the AgCl, and are separated from the silver by the acid parting method.

If copper, platinum and paladium are constituents of the anode, they are dissolved along with the gold, replacing an equivalent of gold in the electrolyte (higher in e.m.f. series than gold), but are not deposited unless allowed to accumulate to high percentage. (Pd must not exceed 5 per cent.)

Each day, gold chloride solution must be added to the cells in order to supply the equivalent of gold which was replaced during electrolysis by Cu, Pt, and Pd. This solution is prepared either chemically or electrolytically. The chemical method of preparation is by dissolving gold in aqua-regia and evaporating off the chlorine, which is an expensive method and is very disagreeable on account of chlorine. The electrolytic method is by electrolyzing concentrated hydrochloric acid between gold anodes, and gold cathodes (in a porous cup), the anode current density 100 amp. per square foot, and the cathode current density 250+ amp. per square foot. Temperature of solution 60° to 70° C. The electrolyte around the anode becomes more and more concentrated in $AuCl_3$. The high current density of the current at the cathode prevents gold being deposited, hydrogen being liberated instead. When the solution in the anode compartment becomes suffi-

ciently concentrated in gold chloride, it is removed and replaced by the solution in the porous cup surrounding the cathode, which in turn is replaced by fresh concentrated hydrochloric acid.

The platinum (and paladium) is recovered from the electrolyte by precipitating as insoluble double ammonium chloride salts $(NH_4Cl)_2PtCl_4$. The gold must first be precipitated by a current of SO_2, the solution is filtered, then NH_4Cl added and the solution evaporated to small volume. The precipitated $(NH_4Cl)_2PtCl_4$ on ignition is reduced to metal, NH_4Cl and Cl being volatilized.

Reactions:

$$2AuCl_3 + 3SO_2 + 6H_2O = 2Au + 6HCl + 3H_2SO_4.$$
$$PtCl_4 + 2NH_4Cl = (NH_4Cl)_2\,PtCl_4.$$

If palladium is present it is precipitated along with the platinum as the double salt. It is separated from the platinum by digesting the ignited precipitate with hot conc. HNO_3, which dissolves the Pd, leaving the Pt as fine particles.

Use of a Pulsating Current for Refining Gold.

It has been found (this journal, Vol. VIII, p. 82) that when a *pulsating direct current* is used for electrolysis of gold anodes, which contain over 6 per cent of silver, the AgCl does not accumulate on the anode and stop the current. Even as much as 20 per cent silver in the anode does not prevent the electrolysis, when using pulsating direct current.

The *pulsating direct current* is accomplished by connecting a single-phase alternator in series with the direct-current generator, the voltage of the alternator being lower than the voltage of the generator, the effect being a pulsating current. The alternating current does not effect the deposition of the gold, but causes the AgCl to break from the anode, thereby avoiding mechanical scrapers for this purpose. The electrolytic effect of the alternating and direct current combination is equal to that of the direct current alone.

The effect of the alternating current on the electrolysis is: (a) it reduces the actual e. m. f. of the cell; (b) it prevents the liberation of Cl at the anode; (c) it breaks the AgCl from the anode; (d) it allows the use of a higher current density; (e) less gold is left in the anode sludge; (f) the electrolysis may be conducted at a lower temperature—25 to 35° C.

The direct current density from the electrodes is 110+ amperes per square foot.

The power consumption is the sum of both the alternating and the direct current.

In the design of the generators, figure on 1.3 volts as the maximum for the direct current generator per each cell in series, and 1.0 volt for that of the alternator per each cell in series. Since the generators are connected in series, their armatures must be so dimensioned as to withstand the heating effect of the maximum total combined alternating plus direct current. The frequency of the alternations has no practical effect on the electrolysis, the usual frequency of 60 per second is very satisfactory.

Electrolytic refining of gold is advantageous in that:

1. It recovers the Pt, Pd, Ir, and Os.
2. It gives the purest kind of gold.
3. It is cheaper than the chemical method of refining.
4. No disagreeable chlorine gas is produced.

Floor of Parting Plants.

The floor of the silver parting plant is laid with an acid-proof and solution-tight material, usually rough concrete bottom with an asphaltum topping. The floor is made with a general slope to a common sump.

At the United States Mints the floors in the gold refinery are made of concrete bottom covered with acid-proof slate slabs two inches thick, with their joints cemented with a mixture of litharge and glycerine. The sweepings from the floor and the residues from the floor washings are melted with the impure gold and cast into anodes and refined.

Cost of Power; Conversion Factors.

By Carl Hering.

Owing to the fact that the cost of power is sometimes stated in cents per hour and at other times in dollars per year, and that the year is sometimes taken at the rate of 24 hours per day and at other times 10, and that this is further complicated by the power being sometimes stated in horse-powers (and in some foreign countries in metric horse-powers) and at other times in kilowatts, the conversions from any one to another sometimes become tedious and are often liable to errors owing to the difference between a unit and a quantity measured in terms of that unit.

The following table of constants has therefore been determined accurately, giving the values of each in terms of each of the others, so that a single multiplication is all that is required for reducing a cost given in terms of one of them into any of the others, as the reciprocals of all values are always given in their proper places. The constants are arranged in the same style and order as those in the writer's "Conversion Tables" for a new edition of which they have been determined. They are in the order of size, beginning with the smallest. The approximate values, which are sufficiently accurate for many calculations, have been selected so as to minimize calculations.

Two kinds of years have been included, one the complete or full year, of 365¼ days of 24 hours each, and the other a working year of 300 days of 10 hours each, and they have been indicated by the names "full" and "working" respectively.

A horse-power-hour or year is really a measure of a quantity of energy and not of power; but as such quantities are popularly referred to as "power" this term has been used in the title, although not strictly correct.

All the values have been carefully calculated by two independent methods besides being cross checked, and it is believed there are no errors.

Cost of Energy; Charges for Power; Cost of Power During Specified Times.

(Money ÷ product of power and time; money ÷ energy.)

Note: A "full" year means 24 hours per day and 365¼ days per year = 8766 hours.
A "working" year means 10 hours per day and 300 days per year = 3000 hours.
Aprx. means within 2%.

1 dollar per kilowatt year (full) [$/kw-yr.]:
= 0.745 650 dol. per hp-year (full). Aprx. ¾.
= 0.342 231 dol. per kw-year (working). Aprx. 34 ÷ 100.
= 0.255 185 dol. per hp-year (working). Aprx. 23/9 ÷ 10.
= 0.011 407 7 ct. per kw-hour. Aprx. 8/7 ÷ 100.
= 0.008 506 16 ct. per hp-hour. Aprx. 6/7 ÷ 100.

1 dollar per horse-power-year (full) [$/hp-yr.]:
= 1.341 11 dol. per kw-year (full). Aprx. add 1/3.
= 0.458 971 dol. per kw-year (working). Aprx. 5/11.
= 0.342 231 dol. per hp-year (working). Aprx. 34 ÷ 100.
= 0.015 299 0 cts. per kw-hour. Aprx. 3/2 ÷ 100.
= 0.011 407 7 cts. per hp-hour. Aprx. 8/7 ÷ 100.

1 dollar per kilowatt year (working) [$/kw-yr.]:
= 2.922 00 dol. per kw-year (full). Aprx. 29 ÷ 10.
= 2.178 79 dol. per hp-year (full). Aprx. 11/5.
= 0.745 650 dol. per hp-year (working). Aprx. 3/4.
= 0.033 333 3 ct. per kw-hour. Aprx. 1/3 ÷ 10.
= 0.024 855 0 ct. per hp-hour. Aprx. 1/4 ÷ 10.

1 dollar per horse-power-year (working) [$/hp-yr.]:
= 3.918 73 dol. per kw-year (full). Aprx. 35/9.
= 2.922 00 dol. per hp-year (full). Aprx. 29 ÷ 10.
= 1.341 11 dol. per kw-year (working). Aprx. add 1/3.
= 0.044 703 8 ct. per kw-hour. Aprx. 9/2 ÷ 100.
= 0.033 333 3 ct. per hp-hour. Aprx. 1/3 ÷ 10.

1 cent per kilowatt hour [cts./kw-hr]:
= 87.660 0 dol. per kw-year (full). Aprx. 7/8 × 100.
= 65.363 7 dol. per hp-year (full). Aprx. 65.
= 30. dol. per kw-year (working).
= 22.369 5 dol. per hp-year (working). Aprx. 9/4 × 10.
= 0.745 650 ct. per hp-hour. Aprx. 3/4.

1 cent per horse-power-hour [cts./hp-hr]:
= 117.562 dol. per kw-year (full). Aprx. 7/6 × 100.
= 87.660 0 dol. per hp-year (full). Aprx. 7/8 × 100.
= 40.233 4 dol. per kw-year (working). Aprx. 40.
= 30. dol. per hp-year (working).
= 1.341 11 cts. per kw-hour. Aprx. add 1/3.

Foreign.

The values of foreign moneys here used are those specified in the writer's "Conversion Tables" and are those used by the U. S. Treasury Dept. They are the true relative values based on gold, as determined by the U. S. Mint in terms of the U. S. gold dollar. The three fundamental values are: 1 pound sterling = 240 pence = $4.8665; 1 mark = 100 pfennige = $0.238; 1 franc = 100 centimes = $0.193. Those constants which depend upon these money values are given to only three or four significant figures; those independent of them are given to six figures, as usual. It has been assumed that the regular horse-power is the only one used in England (as in the United States) and that the metric horse-power is the only one used in the rest of Europe; when this is not the case two calculations will be necessary. One metric horse-power = 0.986-318 horse-power; one horse-power = 1.013 87 metric horse-powers; the difference is therefore about 1½%.

1 franc per kilowatt-year (full):
= 0.811 mark per kilowatt-year (full). Aprx. 4/5.
= 0.735 448 franc per metric horse-power-year (full). Aprx. 22/3 ÷ 10.
= 0.342 231 franc per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.251 693 franc per metric horse-power-year (working). Aprx. 1/4.
= 0.193 dollar per kilowatt-year (full). Aprx. 19 ÷ 100.
= 0.039 7 pound per kilowatt-year (full). Aprx. 4 ÷ 100.
= 0.011 407 7 centime per kilowatt-hour. Aprx. 8/7 ÷ 100.
= 0.008 389 77 centime per metric horse-power-hour. Aprx. 5/6 ÷ 100.

1 mark per kilowatt-year (full):
= 1.23 francs per kilowatt-year (full). Aprx. 1/8 × 10.
= 0.735 448 mark per metric horse-power-year (full). Aprx. 22/3 ÷ 10.
= 0.342 231 mark per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.251 693 mark per metric horse-power-year (working). Aprx. 1/4.
= 0.238 dollar per kilowatt-year (full). Aprx. 12/5 ÷ 10.
= 0.048 9 pound per kilowatt-year (full). Aprx. 49 ÷ 1000.
= 0.011 407 7 pfennig per kilowatt-hour. Aprx. 8/7 ÷ 100.
= 0.008 389 78 pfennig per metric horse-power-hour. Aprx. 5/6 ÷ 100.

1 franc per metric horse-power-year (full):
= 1.359 72 francs per kilowatt-year (full). Aprx. add 1/3.
= 0.811 mark per metric horse-power-year (full). Aprx. 4/5.
= 0.465 337 franc per kilowatt-year (working). Aprx. 7/15.
= 0.342 231 franc per metric horse-power-year (working). Aprx. 34 ÷ 100.
= 0.196 dollar per horse-power-year (full). Aprx. 8/41.
= 0.040 2 pound per horse-power-year (full). Aprx. 4 ÷ 100.
= 0.015 511 2 centime per kilowatt-hour. Aprx. 31/2 ÷ 1000.
= 0.011 407 7 centime per metric horse-power-hour. Aprx. 8/7 ÷ 100.

1 mark per metric horse-power-year (full):
= 1.359 72 marks per kilowatt-year (full). Aprx. add 1/3.
= 1.23 francs per metric horse-power-year (full). Aprx. 1/8×10.
= 0.465 337 mark per kilowatt-year (working). Aprx. 7/15.
= 0.342 231 mark per metric horse-power-year (working). Aprx. 34 ÷ 100.
= 0.241 dollar per horse-power-year (working.) Aprx. 12/5 ÷ 10.
= 0.049 6 pound per horse-power-year (full). Aprx. 5 ÷ 100.
= 0.015 511 2 pfennig per kilowatt-hour. Aprx. 31/2 ÷ 1000.
= 0.011 407 7 pfennig per metric horse-power-hour. Aprx. 8/7 ÷ 100.

1 franc per kilowatt-year (working):
= 2.922 00 francs per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.148 98 francs per metric horse-power-year (full). Aprx. 43/2 ÷ 10.
= 0.811 mark per kilowatt-year (working). Aprx. 4/5.
= 0.735 448 franc per metric horse-power-year (working). Aprx. 3/4.
= 0.193 dollar per kilowatt-year (working). Aprx. 19 ÷ 100.
= 0.039 7 pound per kilowatt-year (working). Aprx. 4 ÷ 100.
= 0.033 333 3 centime per kilowatt-hour. Aprx. 1/3 ÷ 10.
= 0.024 514 9 centime per metric horse-power-hour. Aprx. 1/4 ÷ 10.

1 mark per kilowatt-year (working):
= 2.922 00 marks per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.148 98 marks per metric horse-power-year (full). Aprx. 43/2 ÷ 10.
= 1.23 francs per kilowatt-year (working). Aprx. 1/8×10.
= 0.735 448 mark per metric horse-power-year (working). Aprx. 3/4.
= 0.238 dollar per kilowatt-year (working). Aprx. 12/5 ÷ 10.
= 0.048 9 pound per kilowatt-year (working). Aprx. 49 ÷ 1000.
= 0.033 333 3 pfennig per kilowatt-hour. Aprx. 1/3 ÷ 10.
= 0.024 514 9 pfennig per metric horse-power-hour. Aprx. 1/4 ÷ 10.

1 franc per metric horse-power-year (working):
= 3.973 09 francs per kilowatt-year (full). Aprx. 4.
= 2.922 00 francs per metric horse-power-year (full). Aprx. 29 ÷ 10.
= 1.359 72 francs per kilowatt-year (working). Aprx. add 1/3.
= 0.811 mark per metric horse-power-year (working). Aprx. 4/5.
= 0.196 dollar per horse-power-year (working). Aprx. 8/41.
= 0.045 323 9 centime per kilowatt-hour. Aprx. 9/2 ÷ 100.
= 0.040 2 pound per horse-power-year (working). Aprx. 4 ÷ 100.
= 0.033 333 3 centime per metric horse-power-hour. Aprx. 1/3 ÷ 10.

1 mark per metric horse-power-year (working):
= 3.973 09 marks per kilowatt-year (full). Aprx. 4.
= 2.922 00 marks per metric horse-power-year (full). Aprx. 29 ÷ 10.
= 1.359 72 marks per kilowatt-year (working). Aprx. add 1/3.
= 1.23 francs per metric horse-power-year (working). Aprx. 1/8 × 10.
= 0.241 dollar per horse-power-year (working). Aprx. 12/5 ÷ 10.
= 0.049 6 pound per horse-power-year (working). Aprx. 5 ÷ 100.
= 0.045 323 9 pfennig per kilowatt-hour. Aprx. 9/2 ÷ 100.
= 0.033 333 3 pfennig per metric horse-power-hour. Aprx. 1/3 ÷ 10.

1 dollar per kilowatt-year (full):
= 5.18 francs per kilowatt-year (full). Aprx. 26/5.
= 4.20 marks per kilowatt-year (full). Aprx. 21/5.
= 0.205 pound per kilowatt-year (full). Aprx. 41/2 ÷ 100.

1 dollar per horse-power-year (full).
= 5.11 francs per metric horse-power-year (full). Aprx. 51 ÷ 10.
= 4.15 marks per metric horse-power-year (full). Aprx. 25/6.
= 0.205 pound per horse-power-year (full). Aprx. 41/2 ÷ 100.

1 dollar per kilowatt-year (working):
= 5.18 francs per kilowatt-year (working). Aprx. 26/5.
= 4.20 marks per kilowatt-year (working). Aprx. 21/5.
= 0.205 pound per kilowatt-year (working). Aprx. 41/2 ÷ 100.

1 dollar per horse-power-year (working):
= 5.11 francs per metric horse-power-year (working). Aprx. 51 ÷ 10.
= 4.15 marks per metric horse-power-year (working). Aprx. 25/6.
= 0.205 pound per horse-power-year (working). Aprx. 41/2 ÷ 100.

1 pound per kilowatt-year (full):
= 25.2 francs per kilowatt-year (full). Aprx. 1/4 × 100.
= 20.4 marks per kilowatt-year (full). Aprx. 41/2.
= 4.87 dollars per kilowatt-year (full). Aprx. 2/41 × 100.
= 0.745 650 pound per horse-power-year (full). Aprx. 3/4.
= 0.342 231 pound per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.255 185 pound per horse-power-year (working). Aprx. 23/9 ÷ 10.
= 0.027 378 5 penny per kilowatt-hour. Aprx. 3/11 ÷ 10.
= 0.020 414 8 penny per horse-power-hour. Aprx. 1/50.

1 pound per horse-power-year (full):
= 24.9 francs per metric horse-power-year (full). Aprx. 1/4 × 100.
= 20.2 marks per metric horse-power-year (full). Aprx. 1/5 × 100.
= 4.87 dollars per horse-power-year (full). Aprx. 2/41 × 100.
= 1.341 11 pounds per kilowatt-year (full). Aprx. add 1/3.
= 0.458 971 pound per kilowatt-year (working). Aprx. 5/11.
= 0.342 231 pound per horse-power-year (working). Aprx. 34 ÷ 100.
= 0.036 717 7 penny per kilowatt-hour. Aprx. 11/3 ÷ 100.
= 0.027 378 5 penny per horse-power-hour. Aprx. 3/11 ÷ 10.

1 pound per kilowatt-year (working):
= 25.2 francs per kilowatt-year (working). Aprx. 1/4 × 100.
= 20.4 marks per kilowatt-year (working). Aprx. 41/2.
= 4.87 dollars per kilowatt-year (working). Aprx. 2/41 × 100.
= 2.922 00 pounds per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.178 79 pounds per horse-power-year (full). Aprx. 11/5.
= 0.745 650 pound per horse-power-year (working). Aprx. 3/4.
= 0.080 00 penny per kilowatt-hour. Aprx. 8/100.
= 0.059 652 0 penny per horse-power-hour. Aprx. 6/100.

1 centime per kilowatt-hour:
= 87.660 0 francs per kilowatt-year (full). Aprx. 7/8×100.
= 64.469 0 francs per metric horse-power-year (full). Aprx. 65.
= 30. francs per kilowatt-year (working).
= 22.063 4 francs per metric horse-power-year (working). Aprx. 22
= 0.811 pfennig per kilowatt-hour. Aprx. 4/5.
= 0.735 448 centime per metric horse-power-hour. Aprx. 3/4.
= 0.596 pfennig per metric horse-power-hour. Aprx. 6/10.
= 0.193 cent per kilowatt-hour. Aprx. 19 ÷ 100.
= 0.144 cent per horse-power-hour. Aprx. 1/7.
= 0.095 2 penny per kilowatt-hour. Aprx. 2/21.
= 0.070 9 penny per horse-power-hour. Aprx. 7/100.

1 pound per horse-power-year (working):
= 24.9 francs per metric horse-power-year (working). Aprx. 1/4 × 100.
= 20.2 marks per metric horse-power-year (working). Aprx. 1/5 × 100.
= 4.87 dollars per horse-power-year (working). Aprx. 2/41 × 100.
= 3.918 73 pounds per kilowatt-year (full). Aprx. 35/9.
= 2.922 00 pounds per horse-power-year (full). Aprx. 29 ÷ 10.
= 1.341 11 pounds per kilowatt-year (working). Aprx. add 1/3.
= 0.107 289 penny per kilowatt-hour. Aprx. 3/28.
= 0.080 0 penny per horse-power-hour. Aprx. 8/100.

1 pfennig per kilowatt-hour:
= 87.660 0 marks per kilowatt-year (full). Aprx. 7/8 × 100.
= 64.469 0 marks per metric horse-power-year (full). Aprx. 65.
= 30. marks per kilowatt-year (working).
= 22.063 4 marks per metric hp-year (working). Aprx. 22.
= 1.23 centimes per kilowatt-hour. Aprx. ⅛ × 10.
= 0.907 centime per metric horse-power-hour. Aprx. 9/10.
= 0.735 448 pfennig per metric horse-power-hour. Aprx. 3/4.
= 0.238 cent per kilowatt-hour. Aprx. 12/5 ÷ 10.
= 0.178 cent per horse-power-hour. Aprx. 7/4 ÷ 10.
= 0.117 penny per kilowatt-hour. Aprx. 7/6 ÷ 10.
= 0.087 5 penny per horse-power-hour. Aprx. 7/8 ÷ 10.

1 centime per metric horse-power-hour:
= 119.193 francs per kilowatt-year (full). Aprx. 120.
= 87.660 0 francs per metric hp.-yr. (full). Aprx. 7/8 × 100.
= 40.791 5 francs per kilowatt-year (working). Aprx. 40.
= 30. francs per metric horse-power-year (working).
= 1.359 72 centimes per kilowatt-hour. Aprx. add 1/3.
= 1.10 pfennig per kilowatt-hour. Aprx. 11 ÷ 10.
= 0.811 pfennig per metric horse-power-hour. Aprx. 8/10.
= 0.263 cent per kilowatt-hour. Aprx. 5/19.
= 0.196 cent per horse-power-hour. Aprx. 8/41.
= 0.129 penny per kilowatt-hour. Aprx. 13 ÷ 100.
= 0.096 4 penny per horse-power-hour. Aprx. 2/21.

1 pfennig per metric horse-power-hour:
= 119.193 marks per kilowatt-year (full). Aprx. 120.
= 87.660 0 marks per metric hp-yr. (full). Aprx. 7/8 × 100.
= 40.791 5 marks per kilowatt-year (working). Aprx. 40.
= 30. marks per metric horse-power-year (working).
= 1.68 centimes per kilowatt-hour. Aprx. 1/6 × 10.
= 1.359 72 pfennige per kilowatt-hour. Aprx. add 1/3.
= 1.23 centimes per metric hp.-hr. Aprx. 1/8 × 10.
= 0.324 cent per kilowatt-hour. Aprx. 32/100.
= 0.241 cent per horse-power-hour. Aprx. 24/100.
= 0.160 penny per kilowatt-hour. Aprx. 16/100.
= 0.119 penny per horse-power-hour. Aprx. 12/100.

1 cent per kilowatt-hour:
= 5.18 centimes per kilowatt-hour. Aprx. 26/5.
= 4.20 pfennige per kilowatt-hour. Aprx. 21/5.
= 3.81 centimes per metric horse-power-hour. Aprx. 19/5.
= 3.09 pfennige per metric horse-power-hour. Aprx. 31 ÷ 10.
= 0.493 penny per horse-power-hour. Aprx. 1/2.
= 0.368 penny per horse-power-hour. Aprx. 4/11.

1 cent per horse-power-hour;
= 6.95 centimes per kilowatt-hour. Aprx. 7.
= 5.64 pfennige per kilowatt-hour. Aprx. $4/7 \times 10$.
= 5.12 centimes per metric horse-power-hour. Aprx. $51 \div 10$
= 4.15 pfennige per metric hp.-hr. Aprx. 25/6.
= 0.662 penny per kilowatt-hour. Aprx. 2/3.
= 0.493 penny per horse-power-hour. Aprx. 1/2.

1 penny per kilowatt-hour:
= 36.525 0 pounds per kilowatt-year (full). Aprx. 36.
= 27.234 9 pounds per horse-power-year (full). Aprx. 27.
= 12.500 0 pounds per kilowatt-year (working). Aprx. $1/8 \times 100$.
= 10.5 centimes per kilowatt-hour. Aprx. 21/2.
= 9.320 62 pounds per horse-power-year (working). Aprx. 65/7.
= 8.52 pfennige per kilowatt-hour. Aprx. 17/2.
= 7.73 centimes per metric horse-power-hour. Aprx. 31/4.
= 6.26 pfennige per metric horse-power-hour. Aprx. 25/4.
= 2.03 cents per kilowatt-hour. Aprx. 2.
= 1.51 cents per horsepower-hour. Aprx. 1-1/2.
= 0.745 650 penny per horse-power-hour. Aprx. 3/4.

1 penny per horse-power-hour:
= 48.984 2 pounds per kilowatt-year (full). Aprx. 49.
= 36.525 0 pounds per horse-power-year (full). Aprx. 36.
= 16.763 9 pounds per kilowatt-year (working). Aprx. $1/6 \times 100$.
= 14.1 centimes per kilowatt-hour. Aprx. $1/7 \times 100$.
= 12.500 0 pounds per horse-power-year (working). Aprx. $1/8 \times 100$.
= 11.4 pfennige per kilowatt-hour. Aprx. $8/7 \times 100$.
= 10.4 centimes per metric horse-power-hour. Aprx. 21/2.
= 8.41 pfennige per metric horse-power-hour. Aprx. $1/12 \times 100$.
= 2.72 cents per kilowatt-hour. Aprx. 11/4.
= 2.03 cents per horse-power-hour. Aprx. 2.
= 1.341 11 pence per kilowatt-hour. Aprx. add 1/3.

Recent Progress in Calorimetry.—III.

By W. P. White.

Temperature Measurement and Corrections.

Two previous papers of this series have dealt with the progress that has recently been made in eliminating various calorimetric errors due to uncertain or lagging temperatures in parts of the calorimeter, and, more especially, in its surroundings. To avoid these errors is not difficult; when this has been done the principal errors are in the temperature measurement, and are mainly accidental.

Of *mercury thermometers* for calorimetry the most sensitive are read to 0.001°, but many of the best authorities regard 0.003° to 0.002° as the maximum accuracy with which a temperature interval can in practice be reproduced by one of these thermometers. If somewhat different intervals are employed at different times unavoidable calibration errors will make the probable error appreciably greater. These sensitive thermometers seldom have a range of over 6°. Hence the maximum accuracy which could be expected with them is about 0.4 per mille. This, however, supposes that the full range of 6° is employed. Considerably shorter intervals are often recommended on the ground that Newton's law of cooling cannot be assumed to hold for intervals much over 2°. This recommendation is a mistake. The deviation from Newton's law for a 4-degree interval will not equal the thermometric error (unless the period is considerably over 10 minutes), while, on the other hand, a decrease in the interval will often increase the thermometric error almost proportionally. In fact, the effect of the deviation from Newton's law is usually quite negligible for an 8° interval, or can easily be made so by a rough correction.

The advantage of an increase above 6° is more doubtful with mercury thermometers, since several of the purely thermometric errors would tend to increase with the interval. This remark applies to the substitution of longer range thermometers for the most sensitive ones now in use. *With a given thermometer* it is probably always best, as far as the temperature measurement is concerned, to use the full range. And this has been done, up to 20°, without introducing any error chargeable to the calorimeter or the cooling.

For the best calorimetric work yet reported with mercury thermometers the mean accidental error (that is, the average variation of each determination from the mean result of the series to which it belongs) is about 0.5 per mille.

For electric calorimetric thermometers the sensitiveness and the accuracy can readily be made each 10 times the corresponding value for the mercury thermometer. That is, these electric thermometers can be read to 0.0001°, will reproduce ordinary calorimetric intervals with fair certainty to 0.0002°, and will in general yield measurements agreeing within from one ten-thousandth to one fifty-thousandth (0.1 to 0.02 per mille), their performance in this regard depending mainly on the accuracy of the auxiliary resistances. Two observers, working independently with different types of electric thermometer, have each succeeded in getting calorimetric results whose mean accidental error is one thirty-thousandth (0.03 per mille) or less. The *absolute* accuracy of these determinations may not be greater than 0.2 per mille, but this is sufficient for most present requirements and is as great as the accuracy yet attained in determining the variation in the specific heat of water itself. The electric installation is not only more accurate than a mercury thermometer, but is in some ways more convenient. The observer sitting, notebook in hand, before his galvanometer scale, within easy reach of switches which give complete control of the electrical system, can make readings with great rapidity and in great variety. His facilities in these directions will generally be of more use in the complex and varied problems of the research laboratory than in the simpler and more nearly standardized processes of commercial work. The disadvantages of the electric thermometer are three: The cost, the special electrical knowledge required, and the increased chance of occasional loss of time through slight accidents or blunders in the operation of the more complicated system. None of these disadvantages is really serious, unless the cost is. The strongest reason for not using electrical methods will probably be, in most cases, that greater accuracy than that given by the mercury thermometer is either not needed or is prevented by some cause quite outside the calorimeter, such as impurity or variability of materials.

Two forms of electric thermometer, the resistance thermometer and the thermoelement, have been used in calorimetry and have given results of (so far) about equal and very high accuracy. For some types of recording and regulating apparatus the resistance thermometer is much more sensitive than the thermoelement, but in existing calorimetric thermometers there is little difference in this direction.

The resistance thermometer has at present the advantage that a form of it, developed and now used at the Bureau of Standards, can be purchased ready made, with suitable accessory apparatus. A form accurate at least to 0.0003° in calorimetric work costs, complete with accessories, about $225. For the thermoelement also highly satisfactory accessory apparatus (potentiometers, etc.) can now be purchased, and thermoelements suitable for calorimetry require for their construction neither skill, experience, nor great labor, but they are not regularly manufactured at present. Most standardizing laboratories are better equipped for the calibration of resistance thermometers than of thermoelements, though the calibration of thermoelements is very easy wherever suitable standards are available. The chief advantage of the thermoelement appears to be the greater ease with which the thermoelectric installation lends itself to the simultaneous measurement of several temperatures, and of currents and voltages as well, if desired, but this advantage, again, is greatest in the case of the research laboratory.

The Calculation of Corrections for Cooling seems to deserve attention here because (1) it is mainly through the abbreviation of these calculations that shortening and simplification of the labor of calorimetric determinations can be secured, and (2) the customary formulas for these calculations have been misunderstood to a surprising extent, considering their age and simplicity. It will probably be sufficient, however, to give merely results and conclusions. A derivation of the two formulas here given will be found in the articles on calorimetry already referred to, those in the November and December *Physical Review* of last year. One of the formulas, also, the Pfaundler, has been often presented, though with a bewildering variety of notations, which I have made some effort to simplify.

Three kinds of data may in different cases be needed to get cooling corrections: (1) Data as to the temperature changes produced in the calorimeter by stirring and evaporation; (2) the temperatures of the calorimeter during three periods, namely, the *transfer* period, during which there is communicated to the calorimeter the heat which is to be measured, and two *cooling* periods, before and after the transfer period; (3) the rate of [temperature change of] the calorimeter during the cooling periods.

It is generally known, but sometimes overlooked, that these data may properly be much less accurate than the measurements which determine the main temperature rise, since the cooling correction is only about one-fiftieth of the whole interval. The observer who corrects his transfer period temperatures to thousandths of a degree is simply wasting time. But the rates, though needing no great accuracy in themselves, are each obtained from the difference of two temperatures, so that these temperatures need to be known with considerable accuracy. There may be, therefore, an appreciable error connected with the cooling correction, though this error is not in the cooling itself, but is an additional *accidental thermometric* error, coming in the determination of the cooling rate. Methods of diminishing or eliminating it can more conveniently be presented a little later.

Cooling correction calculations are easier to make if the rate is proportional to the temperature difference of calorimeter and jacket—that is, if Newton's law of cooling holds. The same advantages can be obtained where that law does not hold, however, by introducing the variation from it as a correction. Hence, in what follows, Newton's law will at first be supposed to apply, and the correction for deviations from it will then be discussed.

The simplest case of a complete cooling correction formula occurs where the heat of stirring and evaporation and the jacket temperature are known, for the amounts of direct cooling are then simply proportional to known temperature differences and only one cooling period is needed. For instance: Let ϕ_{tr} be the integrated temperature difference (of calorimeter and jacket) for the *transfer* period (i.e., the period when the heat to be measured is transferred to the calorimeter water), and ϕ_c the same for the subsequent *cooling* period. (The notation here is the same as in the *Physical Review* articles already referred to, except that the subscripts indicating the periods are numbers in those articles.) Let T_{tr} and T_c be the number of minutes in each period, w the heat lost by evaporation and stirring per minute (w, of course, will often be negative), $\delta\theta_c$ the observed temperature fall in the cooling period. Then $\delta\theta_c - wT_c$ is the direct heat loss in period (c), and that in period (tr) equals:

$$(\delta\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c}$$

the cooling correction is, therefore, simply

$$wT_{tr} + (\delta\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c} \qquad (1)$$

This can easily be manipulated into the form:

$$\delta\theta_c + (\delta\theta_c - wT_c)\frac{\phi_{tr} - \phi_c}{\phi_c} + w\,(T_{tr} - T_c) \qquad (2)$$

which is very convenient practically whenever the calorimeter is initially near the temperature of the jacket, for $\phi_{tr} - \phi_c$ will then be relatively small, often very small, while $T_{tr} - T_c$ will often be zero. Hence w, which is always multiplied by one or the other of these, need not be known at all accurately, and the two multiplications will give small products and so can be done by a slide rule, and usually by inspection.

Equation (1) shows how to avoid the thermometric error, mentioned above, which occurs in the determination of the cooling rate. That error comes in $\delta\theta_c$, which is multiplied by $\frac{\phi_{tr}}{\phi_c}$ hence the smaller ϕ_{tr} is, or the larger ϕ_c, the less will be the final error produced. ϕ_c can be increased by lengthening the cooling period, but this method is time-consuming and may tend to introduce other errors. ϕ_{tr}, however, can be decreased by taking the initial temperature below that of the jacket, so that the calorimeter receives heat from the jacket during part of the transfer period and gives it back again during the other part; and the total net cooling (proportional to ϕ_{tr}) is nearly zero. This is not Rumford's old rule of taking the jacket temperature *midway* between the initial and final calorimeter temperatures, but usually calls for a jacket temperature somewhat nearer the final. If this method is not used, and ϕ_{tr} nearly equals ϕ_c [as in formula (2)] the determination of the rate about doubles the maximum possible accidental thermometric error, but, in accordance with the laws of probability, only increases the probable mean error of a series by 42 per cent; hence this method of reducing the cooling correction cuts in two the maximum error, and diminishes the mean error about a third. One price which must be paid for this advantage is an exact knowledge of the value of W, the heat of stirring and evaporation, for errors in this value are no longer diminished as by formula (2).

The method just described, namely, that of diminishing ϕ_{tr}, can be used to save time as well as to gain accuracy. For since it enables us to diminish almost indefinitely the effect of errors in the value of the cooling rate, and, therefore, to get good results with rates inaccurately known, we may determine the relation of rate to temperature (the cooling factor) once for all, and then, assuming that this relation is constant, we may use it in each subsequent determination. We thus get rid of the cooling period altogether, yet with no loss of accuracy. The formula for the correction with this procedure is:

$$wT_{tr} + K\phi_{tr},$$

where K is the constant,

$$\frac{\delta\theta_c - wT_c}{\phi_c} \qquad (3)$$

Some observers have found that the cooling factors of their calorimeters were constant to ½ per cent or better, which suffices for an accuracy of 0.1 per mille even where the calorimeter is started at jacket temperature. The causes which may prevent this constancy do not seem to be fully known as yet, but it seems probable that the measurement of a cooling rate for each separate calorimetric determination will soon be generally regarded as not only a waste of time, but a detriment to accuracy.

Of the labor of getting the cooling correction there now remains only the determination of ϕ_{tr}—that is, the taking of temperature observations during the transfer period and the addition of the results. Even this labor is sometimes avoided, and the value of ϕ_{tr}, the total of the transfer period temperatures, is obtained from the initial and final values of ϕ by means of a table of some sort. The saving in this case is relatively not great, but is often easily obtained. As soon as anyone has performed a few determinations he has the material for expressing ϕ_{tr} in terms of the other temperatures. Of course, this method succeeds better the more uniform the various conditions—such as time, room temperature, material.

In many cases the jacket temperature and the heats of stirring and evaporation are not known. But *if they are constant* they can be eliminated by the use of another cooling period. The most convenient way possible of accomplishing this seems to be by the *Pfaundler formula*. The essential principle of that method is this: The proportionality between heat flow and the temperature difference (ϕ) of calorimeter and jacket cannot be used because ϕ is not known. But one consequence of that proportionality can be made useful, and that is: That the *difference between any two* cooling rates is proportional to the difference of the corresponding temperatures. Hence, not knowing the jacket temperature, we may refer our formula to any other temperature for which we know the rate and proceed much as before.

The additional cooling period furnishes the reference temperature in question. To illustrate, let this period, which comes before the transfer period, and is therefore a preliminary period, be denoted by the subscript, p. Since ϕ is not known, the *mean* temperatures of the periods will be used instead;[1] call these θ_p, θ_{tr}, θ_c. It will also be necessary to use, instead of the temperature intervals, $\delta\theta$, the rates, V_p, V_c, which equal $\frac{\delta\theta_p}{T_p}$, etc. Then the mean rate for the transfer period will be the mean rate for the preliminary period, plus the difference in rate due to the difference in temperature. The cooling correction, which equals this rate, multiplied by T_{tr}, the time of the transfer period, is, therefore:

$$T_{tr}\left\{V_p + \frac{V_c - V_p}{\theta_c - \theta_p}\ (\theta_{tr} - \theta_p)\right\} \quad (4)$$

which is essentially the Pfaundler formula.

Here two things are noticeable. (1) After the first term the formula is exactly like formula (1) in type, but instead of the temperatures of the two last periods there appear the differences between these temperatures and those of the preliminary (reference) period. (2) The heats of stirring and evaporation have vanished, since, being the same for all the periods, they are eliminated in the subtractions. This has always been counted one great advantage of the Pfaundler formula, and is, indeed, evident enough, but still seems to deserve emphasis, for some recent publications indicate that it has sometimes been overlooked.

As ordinarily printed the Pfaundler formula shows a summation, which is the same as the term $\theta_{tr}T_{tr}$ in the form here given. If the periods are all taken equal the computation is easier, since $\delta\theta$ can then be used instead of V. The computation is then as in formula (1), except that the three indicated subtractions must first be performed.

This formula, being like (1), can, like it, be treated as (1) was in getting (2), and thus becomes:

$$T_{tr}\left\{V_c + \frac{V_c - V_p}{\theta_c - \theta_p}\ (\theta_{tr} - \theta_c)\right\} \quad (5)$$

which is very convenient when θ_{tr} is nearly equal to θ_c, since then the computed term is so small that it can be obtained by approximate methods. θ_{tr} nearly equals θ_c when the calorimeter is started at about the jacket temperature, which is usually the most convenient way to do. I believe this variation of the formula is new.

The method described above of diminishing the accidental thermometric error by diminishing the final magnitude of the correction is practically useless with the Pfaundler formula. The proof is simple, and seems unnecessary here. There is, therefore, contrary to a frequently expressed opinion, little or nothing to be gained in accuracy by any particular choice of initial temperature if this formula is used, and the most convenient procedure may be employed, whatever that is. It will often be with the calorimeter temperature near the jacket temperature at the beginning, for which formula (5) is convenient. It also follows that the Pfaundler formula cannot give the highest accuracy, but its compensating advantages are often great. Where the environing temperature cannot be measured, because the environment consists of several bodies at different temperatures, the Pfaundler formula, or something equivalent, is indespensable, and this is probably the case with most existing calorimeters.

On the whole, then, the advantageous treatment of the cooling correction depends upon the observer's knowledge of conditions; the more he knows, the shorter and more accurate his work will be. If jacket temperature and heat of stirring and evaporation are all unknown the Pfaundler formula is essential, with three periods to observe. If these conditions are known the periods may be reduced to two, and accuracy may also be increased by suitable choice of initial temperature. If the cooling factor of the calorimeter is also known the periods may be reduced to one, with no loss of accuracy. If, finally, the form of the heating curve (i.e., during the transfer period) is known, the observations in that one period may be omitted, but, probably, with some loss of accuracy in many cases.

For *adiabatic methods*, also, essentially the same relation holds between the observer's knowledge of conditions and the ease and accuracy attainable. In these methods the jacket temperature, of course, must always be measured, but if the heats of stirring and evaporation are not exactly known two periods are necessary, with the same thermometric error as when formula (2) is used. The periods can be reduced to one, with thermometric error as small as with formula (3), and this requires (as for formula (3)) an exact knowledge of the heats of evaporation and stirring. The cooling factor need not be known for the adiabatic method, but this is not a matter of much importance, since a very approximate knowledge is enough in any case. The computation of the correction is saved in the adiabatic method, but an equivalent labor is substituted for it. In the "diabatic" method the observer reads certain temperatures, which he afterwards adds; in the adiabatic he duplicates these temperatures in the jacket, and must then refill the jacket and adjust it for the next determination.

Graphic methods are sometimes used for computing the cooling correction. Instead of *adding* the individual temperatures, the observer *plots* them, and then measures the plot with a planimeter. There is no question, however, that, with observations equally spaced in time, the easiest and most accurate way, by far, of getting their sum is to add them. The graphic methods are only valuable and, as a rule, have only been used in cases where the observations are taken at irregular time intervals. With such observations all the periods are of irregular lengths, and the labor of computation is considerably increased. The graphic method is merely a means of diminishing the inconvenience inherent in observations of this sort. These irregularly timed observations result from a method of reading which is often adopted with electric thermometers, but which is not in the least necessary; and is even less convenient with properly arranged apparatus. In general, therefore, they can be avoided, and with them the graphic methods which they make necessary.

The treatment of the *corrections for large intervals* remains to be considered. All the above formulas treat the cooling rate as proportional to the temperature difference of calorimeter and jacket, and owe their convenience to that proportionality. It is desirable to use the same formulas even where that proportionality no longer exists. The method of doing this can, perhaps, be best illustrated by the actual description of a simple calculation.

For this method it is necessary to know the difference of calorimeter and jacket, and this involves some measurement of the jacket temperature. But variations in the jacket temperature do not need to be known as accurately as for small intervals. And in the computation only the temperature difference is used; the absolute temperature of the system is of no importance practically.

Suppose, now, in a systematic study of the cooling of the calorimeter, the following results have been reached:

When the temperature difference was:	The cooling per minute was:
1°	0.002°
2°	0.004°
3°	0.007°
4°	0.010°
5°	0.014°

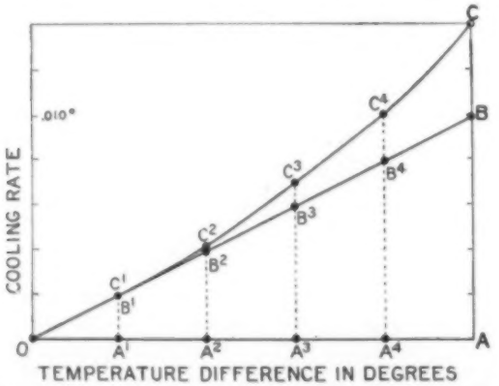
These results are plotted as in Fig. 1, where OA_1, OA_2, etc., represent the observed temperature differences, in degrees, and A_1C_1, A_2C_2, etc., represent the corresponding cooling rates of the calorimeter in thousandths of a degree.

[1] Of course ϕ may be used in the Pfaundler formula if this is desirable.

The straight line OB is now drawn through C_1. The points B_2, B_4, B_5 on it correspond to 3°, 4°, 5°. Since B is straight, and runs through the origin, the vertical distances A_2B_2, A_4B_4, etc., are proportional to th eorizontal distance OA_2, OA_4, etc.

Suppose, now, that a calorimetric determination is made, and that during it observations of temperature difference are made as follows:

1°, 3°, 4°, 5°, 5°, 5°, etc. These numbers have been obtained as a means of afterward finding the cooling that has taken place. That is their only function. It can be performed because to each number there corresponds a particular cooling rate. But a series of numbers *proportional* to the rates, if we had them, would enable us to obtain the rates more easily. Hence, if for the observed differences 1°, 3°, etc., we can substitute another set of numbers which are proportional to the desired rates, the purpose of the original observations will be accomplished and at the same time computation will be facilitated. Such numbers can be obtained from Fig. 1. Thus: the vertical distance A_1C_1 represents the rate for 1°. The ratio between rate and number is 0.002. The rate represented by A_3, C_3 does not bear the same ratio to 3°. But if the difference, 3°, is increased in the ratio $\frac{A_3C_3}{A_3B_3}$, becoming $\frac{7}{6} \times 3°$, or 3.5°, it will bear the desired ratio to the rate, for the rate is 0.007°, which is 0.002 of 3.5°. Similarly 4°, multiplied by the ratio $\frac{A_4C_4}{A_4B_4}$, or $\frac{5}{4}$, becomes 5°, which is proportional to the rate, 0.0100, which prevailed when the difference was 4°. Similarly, 5° becomes 7°, of which the 5-degree rate, 0.014°, is 0.002. And so on. Now, the condition for the



COOLING RATE AS FUNCTION OF TEMPERATURE DIFFERENCE.

cooling correction formulas given above was that rates should be proportional to temperature differences. The new temperature differences which we have just formed artificially fulfil this condition, and so they enable us to use *any* of those formulas with no further trouble.

In practice, the matter is much simpler than in the case just described, which was intended to illustrate the principle. For, (1), the curve of Fig. 1, is not used directly, but the values B_2C_2, B_4C_4, etc., are plotted in another curve on such a scale that they give at once the amounts which must be added to the temperature differences in order to obtain the proportional numbers. Thus, in the present illustration, against the abscissas 1°, 2°, 3°, 4°, 5° would be plotted the numbers 0, 0, 0.5°, 1°, 2°, and a smooth curve drawn through these points. Again, (2), the correction will almost never be needed for calorimetric intervals under 4°, and only occasionally even where the interval reaches 12°. For instance, in one case, where the cooling factor of the calorimeter increased 18 per cent as the temperature difference increased from 1° to 12°, it was nevertheless found that for determinations involving a 12-degree interval the correction for variation of the cooling factor was only $\frac{1}{20000}$ of the calorimetric interval, which, of course, is usually quite negligible. In many cases the total correction would not be negligible, as in this case, but the variations between different values of the correction would be, hence, after a few determinations had been carried through, a single constant value for the correction could be adopted, and no more computation would be needed.

The above method does not require that the cooling factor of the calorimeter remain constant from day to day. It merely supposes that the rates for different temperatures remain in nearly the same relation to each other, which will always be the case. The correction curve may be in error by 16 per cent, and usually much more, without causing perceptible error, as long as it is a smooth curve. But the different numbers obtained from the curve must not show *accidental irregularities* greater than the allowable error of the temperature difference observations, whatever this may be. This condition is easily secured by drawing the curve to a sufficiently large scale. (A discussion of the possible error in this method of correction will be found in the *Physical Review* articles already several times referred to.)

The present paper discusses (1) calorimetric thermometers. Electric thermometers, both resistance and thermoelectric, are at least ten times as accurate as mercurial, and are in some ways more convenient, though more expensive and complicated. (2) Cooling correction formulas are discussed, new forms being given which both cost less labor than the older forms and allow higher accuracy. Adiabatic methods are subject to the same principal limitations as those where corrections are used. In all, the accuracy and the ease increase with the observer's quantitative knowledge of such conditions as jacket temperature, heat of stirring, etc. (3) A method of applying ordinary formulas to large intervals is described in detail.

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Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in May, 1911, was 68. The total tonnage of ore milled during that period was 2,051,975 tons. There were 9,978 stamps in operation with an average duty of 7.66 tons per stamp in 24 hours. Tube mills in commission numbered 218. The yield for the month was 685,951 fine ounces gold. This record is one of the best ever made on the Rand, from the point of view of high tonnage and gross yield. The plant in operation is also greater than ever before.

Chrome iron is used extensively in the manufacture of special steels, either alone or with tungsten, manganese, nickel or other steel-hardening materials. It is also used for lining metallurgical furnaces to prevent excessive corrosion. For this purpose it is used in the crude form or as chrome brick. A third use is in the making of chromium compounds, such as the metallic chromates and alkaline bichromates for use as pigments, mordants and tannages. According to reports from the U. S. Geological Survey, the production of chromite in the United States fell off in 1910, amounting to only 205 long tons, valued at $2,729, as compared with the 1909 production of 598 long tons, valued at $8,300. Almost the entire production of chrome iron ore in the United States has come from California and Wyoming, mostly from the former State. The imports from New Caledonia, Greece and other countries amounted in 1910 to 38,579 long tons valued at $415,768, as compared with 39,624 long tons valued at $460,578 in 1909. Fifty per cent chromic oxide is the basis of valuing chrome iron ores, the price per ton rising or falling with higher or lower percentages.

The Marked Influence of Copper in Iron and Steel on the Acid Corrosion Test*

BY WILLIAM H. WALKER, PH. D.

The unwarranted use which has in the last few years been made of the name of the American Society for Testing Materials in urging the validity of the so-called acid corrosion test, concerning which definite action has been taken in the report of the Committee on Corrosion of this Society at the present meeting, has made the Society unwittingly more or less responsible for two rather grave errors. These are first, that the rapidity of a given sample of iron or steel to dissolve in acid measures its tendency toward natural corrosion, and second, that a low acid test is an index of great purity of the specimen—that is, of an extraordinarily high content of iron. Enough data are now available to show the unreliability of the first assumption, and it is the purpose of this short article to show the fallacy of the second.

In investigating samples of iron and steel which had withstood corrosion for years, and which notwithstanding dissolved in acid very readily, together with samples which withstood solution in acid in a remarkable manner, and yet were rusting at the ordinary rate, it was found that the acid resistant specimens contained in every case a substantial amount of copper; that is, the presence of copper seemed to be the controlling factor in the resistance to solution in acid.

Upon examining the literature of the subject sufficient data to justify this conclusion was found. A number of investigators have studied the matter, and the results of all are fairly well presented in the Carnegie Research Report of Pierre Breuil on Copper Steels in the Journal of the Iron and Steel Institute of Great Britain for 1907. It is here found that small amounts of copper in iron or steel reduce enormously the tendency to pass into solution in acid.

In order to determine whether there is any foundation for the alleged fact so widely disseminated, that a highly acid-resisting iron or steel is also therefore a very pure product, the writer succeeded in having made a number of heats in a basic open-hearth steel furnace, under ordinary conditions, and using ordinary materials, into which metallic copper was added a little before or during tapping. The treatment of the product in the bar and sheet mills, and the annealing operation, was in every case as nearly identical as could be maintained. Although as the work developed many heats were made, the results were so uniform that but few will be here presented.

	C	Mn	S	P	Cu	Loss in weight in 20% sulphuric acid in 3 hours.
A	.08	.50	.018	.017	—	.2235 grams
A_1	.08	.50	.018	.017	.21	.0075 "
B	.10	.41	.027	.026	.19	.0082 "
C	.09	.31	.031	.063	.19	.0095 "

The influence of small amounts of copper in these heats is truly remarkable and shows conclusively that this wonderfully slight tendency to dissolve in acid is not indicative of great purity.

Inasmuch as the presence of metallic copper in contact with iron *increases* enormously the rapidity with which such iron will dissolve in acid, it at first seems surprising that in this case copper should *retard* this action. But we have a direct analogy in the case of zinc and copper. Zinc free from iron dissolves in acid very slowly; mix with it some metallic copper, however, and solution is greatly accelerated. But make an alloy of the two, namely brass, and the solvent action of the acid is negligible. So long as copper is not added to iron or steel in excess of the amount which can remain homogeneously alloyed therewith, the writer can see no reason why there may not be an advantage in its use. But such advantage cannot be ascribed to purity, but rather to what we may call a ferro-brass of small copper content. Reasoning from analogy with zinc, pure iron should dissolve in acid very slowly, but the converse is here shown not to be true, namely, that a slow-dissolving iron or steel is on this account to be considered an exceptionally pure product.

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*A paper presented at the recent meeting of the American Society for Testing Materials at Atlantic City.

Electric Tin Smelting

A Commercial Scale Trial

BY JOHN HÄRDÉN.

Very little has as yet been said or published on the subject of electric-tin smelting; all that has been dealt with in the realm of the electrometallurgy of tin seems to refer to the detinning of tinplate. Stray notices may have appeared in transactions from time to time, but the writer has not been able to find any literature worth referring to, though this may be due to lack of opportunity.*

Even the smelting of tin in the orthodox way—by means of the reverberatory or the shaft furnace—is but scantily mentioned in textbooks, etc., and, as said in a recent editorial note in METALLURGICAL AND CHEMICAL ENGINEERING, very little is generally known, outside the circle of those directly interested in the subject, about the processes of the metallurgy of tin.

It may, therefore, be of interest to metallurgists to learn that electric tin smelting was carried out on a large scale as a commercial experiment in Cornwall, England, during last summer in a plant designed by the author and Messrs. Gröndal Kjellin Company, Ltd., London, as consulting engineers. In the following a brief report will be found of the work carried out, together with a few general remarks on tin smelting, which, of course, are not intended to embrace the whole scope of the subject.

Cornish Tin Smelting.

Tin smelting in the reverberatory furnace has been carried out in Cornwall since time immemorial, and the metallurgy of that process is, in a broad sense, very simple, though to make it an efficient one requires a good deal of minor technical details and long practice. Broadly speaking, the method is this:

The tin ore, chiefly consisting of stannic oxide, SnO_2, as mined in Cornwall, is being dressed in streams or on tables in the mines, so as to contain about 63 per cent to 64 per cent of tin. The ore is delivered in bags of about 3 cwt. each in the form of a dense, red-brown, moist powder, fairly finely ground, say about 15 to 20 mesh. After allowing to dry for some little time (depending upon the moisture, which may vary from 7 per cent to 15 per cent), the ore is mixed with the reducing agent—a low-grade anthracite called "culm"—which contains a considerable quantity of slatey clay or aluminium-magnesium silicate, etc.

The quantity of culm to be used depends, to a certain extent, upon the character of the ore. It is unusual to use one kind of ore only; indeed, the proper blending of different ores is one of the principal factors in producing an efficient melt. Bolivian or Straits tin ore is very often mixed with the Cornish ore. Generally about 20 per cent to 25 per cent of culm is used; for the first charges no other flux is added. One furnace is generally built so as to take a charge of three to four tons of ore and about three-quarters of a ton of culm; a certain portion of the culm acts as fuel. A quantity of the ores is previously mixed with culm in various proportions and smelted in a test crucible of graphite so as to give best yield.

It is very essential to conduct the smelt so that the temperature is raised gradually, especially if the ore is of a silicious nature, because if the temperature rise is too brisk tin silicate and silicides will be formed, which will go into the slag directly,

*See also: Reduction of Tin Dross in an Electric Furnace, Am. El. Ch. S., 1910, by R. S. Wile.

causing a great loss. Also volatilization must be guarded against, as the metal is comparatively strongly volatile. In more modern plants a condensing chamber is used to recover the losses. The charge is frequently rabbled with iron rakes so as to ensure an even distribution of the heat. In some plants a quantity of scrap iron is added to the charge before the melt is finished in order to cause impurities to be taken up by the slag rich in iron which is being formed.

The melt is finished after six to eight hours, when the taphole is opened and the metal and slag are run off together into a hemispherical cast-iron pan, which is often provided with special heating arrangements. Each pan will hold four to five tons of metal. The slag floats on the surface of the metal and is run off into iron boxes to solidify.

This first slag is generally rather rich in tin; it may sometimes contain 14 per cent to 16 per cent of tin in various combinations. It is, therefore, frequently used as an admixture to a subsequent smelt, or may be treated separately for tin. The final slag is said to contain about 6 per cent to 7 per cent of tin, and is, therefore, sometimes smelted together with lead refuse for solder tin. In more modern plants, however, the slag contains only 2.5 per cent to 3 per cent of tin.

The crude tin, as first obtained, is cast into large ingots, still containing a considerable quantity of impurities. These ingots are placed on the sole of the reverberatory furnace and remelted at slow heat and low temperature, so-called liquation process. The pure tin is thereby run off and the impurities remain behind, forming the so-called "hardhead," which chiefly consists of an alloy of tin with sulphides and arsenides of iron. The percentage of tin varies somewhat, but is always considerable, and this "hardhead" is causing the smelter considerable trouble. One of the objects of the electric smelting was to try to eliminate, if possible, the formation of this undesirable hardhead, which is so difficult to treat separately.

After the tin has been liquated it is collected in a large pan, or refining pot, heated from underneath. This tin is usually about 96 per cent to 98 per cent pure and still contains impurities of arsenic, tin-suboxides, etc. For further purification so-called "tossing" and "boiling" are employed. The former consists in scooping the hot metal in thin broad streams through the air, whereby the impurities are oxidized and float on the top. The "boiling," which is more common, consists in immersing a raw piece of wood of large dimensions in the metal, vigorously stirring the same. (Formerly it was believed that only wood from apple trees would serve the purpose properly.) A violent agitation sets in, oxidizing the impurities.

In modern plants dry steam and air are blown through the metal for the same purpose.

A small sample is ladled out from time to time and poured into the groove of the "test stone," a porous, white sandstone; the behavior of the solidifying metal, together with the fracture of the notched bar and the special sound of the "tin crying," tells the skilled smelter of the purity of his metal. Indeed, this test is so marked that an experienced smelter can tell within 0.2 per cent of the purity of the tin. Refined tin should have 99.6 per cent to 99.8 per cent of tin.

The refined tin is usually cast into 28-lb. ingots and branded with the trademark or else into strips of 18 in. and about ½ in. thick.

Electric Tin Smelting.

At a first glance it would seem that the electric furnace would not adapt itself very well to the smelting of tin ore, since the temperature of the electric furnace would appear to be unnecessarily high for the reduction, and even injurious. However, this assumption is quite erroneous. The reduction of stannous oxide by carbon requires a comparatively high temperature, especially if a good separation from impurities is to be effected. Besides, tin is readily volatile at this high temperature and in a reverberatory furnace with its wide hearth communicating directly with the smoke stack or else in the tin blast furnace where air is forced through the charge this volatilized metal is mostly lost altogether if complicated condensing devices are not provided for.

All this is mostly avoided with the electric furnace, if properly designed. Another feature is that the so objectionable formation of "hardhead" can be greatly reduced if the process is carefully conducted, as will be seen later. Finally, if the cost of power is not too high a decided economy, compared with the reverberatory furnace, is quite possible, as the experiments made have shown.

As to the losses of tin in slag, it has been conclusively shown that one has it in hand to so conduct the operation that slags containing only 0.25 per cent of tin may be produced in one melt; but it appears to be more economical to so conduct the process that a richer slag, containing, say, 14 per cent to 16 per cent of tin, is first produced, and to treat this richer slag while still hot in a second furnace as in this case the output is very much increased.

Theoretical Yield.

If tin oxide, SnO_2, is reduced by means of carbon, we may assume that the ideal equation is

$$SnO_2 + C = Sn + CO_2 \qquad (1)$$

but that part of the reaction takes place according to the equation,

$$SnO_2 + 2C = Sn + 2CO \qquad (2)$$

This is substantiated by the fact that a considerable quantity of CO can be seen burning at the mouth of the furnace; a certain amount of the CO will naturally serve as a reducer when passing through the charge piled up in the shaft, and, for reasons given before by the author in his article on "Electric Iron Reducation" (see Electrochemical and Metallurgical Industry, Vol. VII, 1909, page 16) it is of importance to so conduct the charge that the reaction (2) is predominant, though it will hardly be possible to realize this in practice to a greater extent than 50 per cent. Also the packing of the charge in the shaft, owing to the heaviness and fineness of the ore, will prevent this to a certain extent. We may therefore presume that under proper conditions, and after some hours' run, the reaction goes on according to 50 per cent of each of the equations 1 and 2.

The Heat Energy Required.

According to Thomson and other investigators, 118 grams of tin (one gram-molecule) will yield 145,300 gram calories by combustion into tin oxide. This amount of energy will, therefore, be required for the liberating of 118 grams of metallic tin.

On the other hand, the combustion of 12 grams carbon into CO_2:

$$C + 2O = CO_2$$

will yield 96,960 gram calories,

and the combustion of 24 grams of carbon into 2CO:

$$2(C + O) = 2CO$$

will give (in round figures) 58,000 gram calories.

Assuming, therefore, that the reaction takes place according to equation (1), or $SnO_2 + C = Sn + CO_2$, we can calculate the energy requirements of the reaction per ton of metallic tin as follows: 145,300 — 96,960 = 48,340 kg calories per 118 grams of tin, or 409,661 kg calories per kilogram of tin, and as one kw-hour = 864 kg calories we have for one ton of tin:

$$\frac{409{,}661 \cdot 1000}{864} = 474 \text{ kw-hours per ton of tin.}$$

In the same way we find that, if the reduction takes place according to equation (2) or $SnO_2 + 2C = Sn + 2CO$, 855 kw-hours per ton of tin are required.

Assuming that the reaction takes place to the extent of 50 per cent of either of the reactions (1) or (2), around 665 kilowatt-hours per ton of tin are required.

In addition to this, a certain amount of energy is required for the heating up of the charge to the temperature of reaction. No actual measurements of the temperature in the furnace were taken, but, judging from the appearance of the slag and the

metal tapped, it may be safe to assume that the main average of the temperature of the whole interior of the reactive zone was about 1400° C. to 1600° C.

Assuming that the ore contains 90 per cent tin oxide, we require 1280 kg of ore for each ton of tin. Experience has shown that about 16 per cent to 20 per cent of slag per ton of ore is an average, or about 220 kg of slag per ton of tin. The melting of 1 kg of slag requires, as an average, 500,000 gram calories = 0.6 kw-hour, or around 130 kw-hours for each ton of tin.

The heat capacity of the charge and the molten tin we may set at 65 kw-hours as being on the safe side.

The radiation losses through the furnace walls, electrodes and water jackets are naturally of such nature that no accurate figures can be obtained, but it seems safe to assume this loss to be equal to that consumed by the slag, or 130 kw-hours. The escaping gases, in the furnace used for our experiment, left the furnace mouth at a temperature of around 800° C.; the losses through this source may therefore be set at about 150 kw-hours for each ton of tin.

In summing up these items we therefore find that in practice it should be possible to produce one ton of tin with an expenditure of power equal to:

For the reaction	665	kw-hours
For the slag	130	"
For the specific heat	65	"
For the radiation losses	130	"
For the losses through gases	150	"
		"
	1,140	"

In the experiments carried out this theoretical figure naturally could not be obtained, as the furnace was too small, and also for other reasons given below, but a very fair result was obtained, as will be shown.

Practical Tests.

At the request of Mr. A. F. Maclaren, of London, the author was given the task of carrying out some experiments on these lines late in 1909. Some difficulty was at first experienced in finding a suitable place for the preliminary trials on a laboratory scale, but early in 1910 Dr. Walmsley, of the Northampton Institute, courteously permitted the tests to be made in the Electrochemical Laboratory of the Institute. A furnace had been designed for 800 amp and 50 volts alternating current, though only direct current of about 300 amp and 100 volts was obtainable at the Institute. The direct current is especially unsuitable for this class of work on account of the heating effect being more prominent on the positive electrode, which was fully manifest during the operation. Also electrolytic effects on the slag were to be reckoned with. In addition to this, the tests must be carried out so as not to disturb the ordinary work of the laboratory, which naturally also hampered the results.

In spite of these difficulties the following results were obtained, which proved the feasibility of the matter:

The furnace was built up in a manner somewhat resembling the one designed by Harmet for iron ore reduction. It was provided with a bottom chamber of magnesite bricks, rammed with calcined magnesite and tar, in the usual manner. A charging shaft rested on a vault or arch; the inner diameter of the hearth was about 5½ in. and the total height about 14 in.

Two lateral and inclined carbon rods about 2½ in. in diameter round were used as electrodes and reached into the center of the hearth about 2½ in. from the bottom. No cooling jacket was used, though it would have been of great assistance.

The wear and tear of these electrodes was so little, in spite of the primitive arrangement, that it could hardly be taken into account for the experiments; this was also manifested during later experiments.

A preliminary heat was made on Jan. 27, 1910. The charge consisted of

Tin ore (about 63 per cent of tin)	20	lb.
Coke powder	5	"
Flux	5	"

Of this only 12 lb. of ore was actually reduced owing to lack of power. Totally about 40 kw-hours were spent, of which 15 were used for heating up and 25 for the reduction proper. The smelt yielded 5¼ lb. of pure bar tin and 1¼ lb. in shape of shots, thus corresponding to about 85.5 per cent of yield from ore. The slag was very irregular, as only one mutual taphole was provided for, so that some metallic tin remained in the slag. Thus the power expenditure amounted to around 4 kw-hours per pound of metal, or 8960 kw-hours per ton. Great losses of heat were experienced owing to the primitive arrangement of the furnace.

In the next trial the slag from the first melt was used as a flux. This proved to be a disadvantage, as it required a much higher temperature to be remelted, and the taphole froze very readily.

In a subsequent test 16 lb. of ore were smelted, giving 6½ lb. of pure bar tin and 2½ lb. of shots, or a total of 9 lb. of metal, equal to around 90 per cent of the metal in ore. The maximum power was 8.4 kw and the total duration of the melt two hours. The total power used, including heating up, was 19.2 kw-hours, or 2.15 kw-hours per pound, or about 4900 kw-hours per ton of metal.

The tin obtained direct from the furnace proved to be well over 99 per cent pure; it contained very little iron and no arsenic. The slag varied in appearance; if the run was smooth it flowed freely and showed a grayish glassy luster. Three average samples of this proper slag were analyzed and found to contain in the average 0.5 per cent of tin. The stiff black slag obtained in some melts contained, however, as much as 8 per cent of tin. Thus the best efficiency obtained from this little furnace, thermically, was about 25 per cent, while the metallurgical yield was as high as 90 per cent.

A test for electrolytic recovery of the tin in the slag was also made. Some slag and refuse containing an appreciable amount of metal was fused with soda (which may be carried out at once while the slag is hot). This was readily soluble in water and could be electrolyzed direct, with iron plates as cathode. A good deposit of tin was obtained, though the efficiency was not very high.

If, however, the oxide from the slag is converted into chloride of tin the theoretical power consumption would be about 1820 kw-hours per ton of metal. Assuming an efficiency of, say, only 50 per cent, this recovery may be possible, where the power price is low, with a rotating cathode and scrap tinplate as anode.

In view of the above promising metallurgical results it was decided to carry on experiments on a much larger scale on a commercial basis. A power supply of 100 kw was secured and a suitable site acquired where a galvanized iron building was erected, a tank for cooling water was built and every facility provided for a conclusive test.

The Electrical Equipment.

Current was obtained from a Diesel engine plant supplying power for a tin mining concern; the condition was that no disturbance or great fluctuation should be caused which might endanger the regular working of the pumps in the mine. The electric-power supply was three-phase, 50 cycles, 650-675 volts.

To meet this requirement a Berry three-phase transformer, oil-cooled, was installed, receiving constantly 660 volts on the primary side, in star connection, with only lightning arrester and main switch on the line; the secondary side, which was also in star connection with earthed neutral, had four tappings capable of giving either 30 volts, 40 volts, 50 volts or 60 volts. The terminals ended in copper busbars on a slate panel provided with heavy copper screw plugs for obtaining the desired combinations. Each phase on the secondary side was provided with an ammeter; a voltmeter with a change-over switch served for

all three lines, while the recording and indicating wattmeters were on the primary side.

The busbars were connected with the furnace electrodes by means of each four flexible cables of suitable size arranged so as to give the least possible induction. As, however, the current was as high as about 2500 amp, a certain drop in power factor was inevitable; it was on the average about 0.90.

During the whole trial period no disturbance whatever was caused by the electrical gear, though one or two thunderstorms were experienced; the distance between power house and smelting plant was about 600 yd.

The Furnace.

The furnace used is a shaft furnace of special design. Owing to pending patent applications a detailed description of this, as well as of the details of the actual smelting process employed, will have to be deferred for a later occasion. One of the principal features in an ordinary reverberatory tin furnace is that the metal, like mercury, will percolate into the lining and brickwork in large quantities and remain lodged there until the furnace is broken up. In a large plant with a number of furnaces this means a locking up of a considerable capital for a long period, which, naturally, is objectionable. This is successfully avoided in the new design of the electric furnace, where only a minimum quantity of metal is withheld, all the metal being drawn off as soon as it is formed.

Three electrodes of carbon 8 in. x 8 in. square and 4 ft. 6 in. long, cooled by water jackets, were used. The electrodes were held by means of gripping holders extending over half the electrodes and moved upward as the ends are worn off. The holders were suspended by means of flexible steel rope and worm-gear winches, permitting of a fine adjustment.

No direct arc was permitted to be formed, as this caused a great variation of the yield and the working of the furnace. The charge formed a cone-shaped heap in the hearth, covering the reacting zone, permitting the electrodes to "burn free," forming a small space filled with incandescent gas round the ends which acted as a resistance, in which the reaction took place. This is, as is well known among furnace men, the only efficient way of properly working a furnace.

Arrangement was made to prevent the escaping of volatilized metal; only about 0.5 per cent or less of tin found its way out of the furnace.

Tapholes on different levels were provided for drawing off metal and slag separately. Sometimes the slaghole may be frozen up with a hard slag crust, especially if the ore is rich in iron which all went into the slag. In that case a small auxiliary electrode was used for burning the taphole open again, which could be made very easily, though it should be admitted that an oxy-acetylene jet would be even more handy.

The Smelting Process.

At the start of a campaign, which usually lasted 10-12 days, on account of the experimental nature of the trials, though there was no objection in keeping the furnace working for months in succession, the empty furnace was first pre-heated for about four to six hours by means of a small wood and coke fire in order to prevent cracks in the brickwork. In subsequent smelts this could be almost entirely dispensed with, since the lining had properly set.

A charge of about 14 kg of culm for each 100 kg of ore, depending somewhat on the nature of the ore, was gradually filled in, while a current of about 1000 amp per phase, at 60 volts, was turned on. As the reaction commenced to set in the volts were lowered gradually to the working condition of 40 volts and 2500 amp, which was steadily maintained during the whole run. Only on special occasions, with certain ores, was 30 volts used.

The current fluctuated considerably at the beginning as the cold charge was being thrown in, but as soon as the electrodes had commenced to "burn free" properly the fluctuations ceased. Already half an hour after the first charge could the first tapping of metal take place, while the first slag was drawn after the furnace had been in proper working for a couple of hours. The appearance of the slag rendered it possible to the experienced eye to judge if the charge was of right composition or not; if of a glass-like, dark brownish green color, and freely leaving the taphole, the charge was right; if sluggish and light brown it had to be altered.

After a few hours' run, certain reagents were added at regular intervals; this successfully prevented the formation of "hardhead," causing the sulphides and arsenides of iron to enter into the slag instead of alloying with the tin. The result was that the metal, as it left the taphole of the furnace, was about 98 per cent pure, if the right blend of ore, not too much contaminated with impurities, was used. In one campaign ordinary Bolivian ore only, with only 49.5 per cent tin, and about 15 per cent of iron, was smelted solely for about ten days, and yielded over 92 per cent totally, on some days even 97 per cent of the metallic tin from the ore.

The liquid metal was accumulated in the usual way in large pots of wrought iron, heated from underneath and the sides. By means of an air-pump air was blown through the metal through a perforated iron pipe, whereafter the metal was ladled into a second tank to set properly. With this simple operation a metal of 99.75 per cent purity, and even more, was obtained, though this is well up to commercial standard.

The metal was then poured into ingots of about 28 lbs. weight, branded and stored in the usual way. The accompanying

STOCK OF INGOTS OF ELECTRIC TIN.

photograph shows a stock about 2 tons of ingots, being produced in about 1½ days.

Output, Power Consumption and Loss in Slag.

All three of these factors were found to be in close relation to each other. It was thus found that a slag containing as low as 0.25 per cent of metallic tin could easily be produced, but to do this in one operation proved to be less economical with the prevailing power price. The output was only about 6 to 8 cwt. per day, and the power consumption correspondingly high, or about 3000 kw-hours per ton of metal.

Going to the other extreme, the operation could be so conducted, by altering the composition of the charge, that a slag containing 17 per cent to 19 per cent of tin was produced. In that case the output increased to 1 ton 11 cwt. per day with a power consumption of 1300 kw-hours per ton of metal.

As only one furnace was provided for, this manner of operation was not commercial, with such high losses of tin in the slag, although this slag could be treated again separately. It was found most economical under the prevailing conditions to operate the furnace so as to obtain a metallurgical yield of 96 per cent to 98 per cent, with a higher power consumption. The following weekly report for the week July 28 to August 4 is fairly typical:

Ore used: 21,056 lb. @ 57 per cent	=	12,003	lb. metal
Scrap metal and charge refuse....	=	2,324	" "
		14,327	" "

Tin produced, 14,170 lb.
Net yield, 98.75 per cent.
Total number of kw-hours, 15,113.
To be deducted: for slag treatment (experiment), 678; for preparing furnace, etc., 500.
Total number of kw-hours used for reduction, 13,935.
Units per ton = 2220.
Electrode consumption, 28 lb. per ton of metal.

It should be noted that this figure for the power consumption was taken as an average over a single run of eight days, during which experiments with different charges were made. On single days the power consumption was only 1700 kw-hours per ton.

Conclusions.

The above figures bear out that a considerable saving in tin can be effected by means of electric smelting. If the plant is so arranged, as is so often done in other metallurgical processes, that two furnaces are working in series, the first one providing metal at a great rate continuously, and permitting the hot slag, rich in tin, to run direct into the second furnace for subsequent treatment, eventually with lead ore for solder in order to hasten the process, the power consumption should not exceed 1400 kw-hours per ton of tin (or about 55 per cent of theoretical) tin, in a sufficiently large furnace.

Even this figure may be decreased if proper blending of ores is strictly reverted to. In the above-mentioned cases single ores were mostly used, on account of the high price of the ore, which prevented the laying in of a large stock of expensive materials for experiments.

Also, it should be remembered that in a large commercial plant all the modern appliances for tin smelting should be applied, which again would help to increase the yield and lower the cost.

Another feature is of importance: The electric furnace will commence to yield pure tin already during the first hour of operation, and continue to do so without interruption, while the reverberatory furnace is intermittent in its operation.

Further, the formation of "hardhead" is materially minimized.

The amount of carbon necessary for the reduction is only 14 per cent, instead of 20 per cent to 25 per cent. Less labor is required, as three ordinary laborers per shift are quite capable to run the furnace.

Much less space is required for the electric furnace than for the reverberatory.

The consumption of electrodes is comparatively so small that it need not be regarded as an objectionable feature anywhere.

If the power can be obtained at a reasonable rate it therefore appears to be quite feasible to smelt tin electrically; especially should this be the case in the Straits Settlements, where large quantities of good ore, but not very rich in tin, are mined, and where waterfalls are said to exist in abundance, but only low grade of coal, and in small quantities. The use of charcoal, which probably would be much cheaper, should give no objection whatever.

In fact, with suitable flux, charcoal would probably work even better than culm.

Native labor could be employed under white supervision without hesitation, as practically no skilled labor at all is required. It is obvious that it would be much cheaper to ship metallic tin than an ore, which, as a rule, contains only about 55 per cent of tin. Naturally the working would at first cause a considerable amount of trouble, and would require a good deal of metallurgical and chemical engineering, but, on the whole, it seems justifiable to say that the principles of the problem are solved.

The above experiments will probably be resumed in a short time, in a somewhat modified way, by which an even greater saving is reasonably expected.

London, England.

The Right of Employer and Employee in Questions of Priority.

A recent decision in an interference case rendered by the United States Patent Office goes deeply into the legal right of employer and employee in questions of priority of invention. It is therefore of such a general and broad interest that we give here its salient features.

The case at issue was the following:

Davis E. Parker filed on May 18, 1906, serial No. 317,516, a patent which was granted him Jan. 5, 1909, bearing No. 909,017, covering an apparatus for preparing oxone.

An application covering practically the same subject was filed on Feb. 17, 1909, serial No. 478,508, by George Fred. Brindley, who was acting at the time in the interest of the Niagara Electro Chemical Company, Niagara Falls, N. Y.

To better illustrate the case we give herewith extracts of the decision rendered on June 9 a. c., which will give a clear understanding of the same, without going deeply into technical details of interest only to those immediately affected in the matter.

"This is an interference proceeding in which the subject-matter relates to an electrical furnace for fusing sodium peroxide. The fused peroxide has been given the trade name of 'oxone.' The issue, which is drawn to the details of the furnace structure, is defined in five counts . . . .

"The senior party, Parker, is a patentee, his patent No. 909,017 having been issued on Jan. 5, 1909, on an application filed May 18, 1906. Brindley's application was filed after the grant of the patent to Parker and is substantially a copy of said patent and the subject-matter of the interference is therefore the five claims of the Parker patent. The question at issue is one of originality rather than the ordinary one of priority of invention. Each of the parties has taken testimony in an effort to prove his claim as the originator of the invention in issue.

"At the time the invention in issue was originated both of the parties were in the employ of the Niagara Electro Chemical Company at Niagara Falls, N. Y. Brindley was works manager of the company and is shown to have had considerable training and experience in chemistry and metallurgy. He left the employ of the company in May, 1906. Parker was a machinist and was in charge of the machine shop of the company. He entered the employ of the company on March 28, 1904, and left in November, 1905. While both parties were in the employ of the same company, Parker's position was that of an assistant or subordinate to Brindley and the relation of the parties was therefore such as to bring them within the principles of employer and employee. In fact, there is no contention that the doctrine of employer and employee is not applicable to the case.

"The record shows that experiments for the production of fused sodium peroxide were begun by the Niagara Chemical Company at least as early as June, 1904. Brindley took up the problem at the Niagara Falls plant and during 1904 tried various kinds of apparatus. He claims to have built and tested a furnace like that shown in his Exhibit No. 1, and Parker admits having done the machine work on this furnace."

Then follow numerous quotations from the voluminous testimony taken in this case and the examiner then writes:

"These examples clearly indicate an indifference to accuracy on the part of Parker which cannot help detracting from the credibility of his testimony in a controversy of this character. It follows, therefore, that the question at issue will have to be determined largely from the circumstances of the case and the admissions of the parties together with the few facts that are not disputed or which are clearly established by the record."

The examiner goes thoroughly into the records submitted and comes to the following conclusion:

"Granting, therefore, that Parker did originate some of the features, it is not believed that he is entitled to claims covering a combination of such features with features which were admittedly originated by Brindley. In other words, if Brindley originated some of the features of the claim and Parker others, it seems clear that some claims must be regarded as the invention of Brindley and not of Parker. In fact the weight of decisions is believed to be clearly to the effect that such claims should belong to the employer rather than the employee. It is true that when an employee makes an invention which is independent of disclosures made to him by the employer and is able to prove his claim he and not the employer is entitled to the invention (Robinson v. McCormick, 128 O. G. 3289, 29 App. D. C. 98). But in no case known to this tribunal has it been held that claims to a combination made up of features admittedly furnished by an employer and others by the employee should be regarded as the property of the latter. In fact, the courts have frequently held that even where the features contributed by an employee amount to an independent invention a presumption exists in favor of the employer which can be overcome only by satisfactory proof that the independent invention was originated by the employee. On this point the Court of Appeals has said:

"What we hold is that when in the course of experiment by an employer with an invention a device is suggested for its improvement which in itself would reach the dignity of independent invention, and a dispute arises between employer and employee as to its conception, the presumption is justly in favor of the employer, and it is incumbent on the employee to overcome the presumption by satisfactory proof." (Miller v. Kelley, 96 O. G. 1038; 18 App. D. C. 163.)

"Evidently, therefore, in a case like that at bar where some of the features admittedly originated by the employer and the origin of the other features is in dispute a presumption exists that such other features were the work of the employer rather than the employee. And this presumption is strengthened when the particular facts and circumstances of the present case are considered. Brindley was admittedly the first to take up the problem of fusing sodium peroxide. He developed various forms of furnaces for this purpose before anything is claimed to have been done on the case in issue. He made the drawings (Brindley's Exhibits Nos. 3 to 10) which led up to the furnace in issue. He determined the size of the various parts which entered into the completed furnace, gave the order to build the furnace and, in fact, appears to have directed everything concerning the actual preparation of the first furnace involving the invention. Practically no evidence except the testimony of Parker himself has been introduced to show that any of the details were originated by Parker and this is not sufficient to overcome the presumption in favor of Brindley.

"Considering all the facts and circumstances of the case, it is believed that evidence presented on behalf of Brindley is sufficient to overcome the burden of proof under which he is placed and to show that the invention in controversy originated with him rather than with the senior party, Parker.

"On behalf of Parker it is contended that bar of public use exists the grant of a patent to Brindley and it is requested that, without deciding the question of priority, the examiner of interferences direct the attention of the commissioner to this fact. There appears to be no authority for the action requested. In fact, it appears to be well settled that where testimony has been taken and a case is in condition for final disposition action will not be suspended for the purpose of instituting public use proceeding (Brenizer v. Robinson, 166, O. G. 1281; Wert v. Borst and Groscop, 122 O. G. 2062; Doble v. Henry, 118 O. G. 2249). The question of priority has therefore been considered.

"The drawings made by Parker in the latter part of 1905 and the testimony relative thereto have not been discussed, as these matters appear to have little bearing on the question of priority. Likewise no reference has been made to the testimony bearing on the question of public use, although this testimoney has been considered. It is not believed, however, that the circumstances of the case are such as to justify a recommendation as to public use under the provisions of Rule 126.

"Priority of invention of the subject-matter in issue is awarded to George F. Brindley, the junior party."

The case has been appealed.

Notes on Technical Analysis.

By R. S. Davis.

Analysis of Wulfenite Ores.—The Wulfenite ores found in the Southwestern States and Mexico usually contain more or less tungsten and vanadium, and as these elements exert a decided influence on the metallurgical process of extraction a practical method for the analysis of such ores was devised by the writer, as follows:

Pb. Mo. V. Treat one gram of the agate-ground sample with 10 cc. HCl and 5 cc. HNO_3, warm on hot plate and after the action has comewhat subsided add 10 cc. H_2SO_4 and evaporate to strong sulphuric fumes. Cool somewhat, dilute and heat until all is in solution excepting the lead sulphate and silica. Filter into a pressure bottle (a citrate of magnesia bottle with patent clasp stopper is well adapted for this purpose), washing with water containing a little sulphuric acid. Dissolve the lead sulphate in hot ammonium acetate and acetic acid, filter from the silica, washing well with hot ammonium acetate solution and estimate the lead by titrating with ammonium molybdate. If desired, one may add a large excess of sulphuric acid to the lead acetate solution, boil, add alcohol and determine the lead gravimetrically as sulphate.

Pass H_2S into the filtrate contained in the pressure bottle for about fifteen minutes, then stop the bottle, place in a vessel of warm water and heat to boiling for twenty minutes. If the amount of MoS_2 is small, and a preliminary test has shown copper to be absent collect the precipitate in a Gooch crucible, dry and ignite gently until there is no sulphurous odor, then heat to faint redness and weigh as MoO_3, containing 66.57 per cent. Mo. If this molybdic acid is not entirely soluble on warming with ammonia the residue should be deducted. It is safer, however, especially if there is much molybdenum, to proceed as follows:

Dissolve the MoS_2 in HNO_3, evaporate in a casserole to dryness then heat over a wire gauze to remove all sulphuric acid. Add 5 cc. ammonia and warm until the MoO_3 is all in solution (if copper is present use KOH instead of ammonia, dilute, boil, filter off the cupric oxide and estimate the copper by any convenient method). Filter if necessary, acidify with acetic acid, boil and add an excess of lead acetate. Filter and wash the precipitate with hot ammonium acetate solution to remove any trace of lead sulphate then with hot water, ignite gently and weigh as $PbMoO_4$, containing 26.15 per cent. Mo. The writer has not had much success in estimating Mo by the zinc reduction processes excepting when very little Mo is present.

Boil the filtrate from MoS_2, which will be green or blue if vanadium is present, to remove H_2S. Then add permanganate to a pink color. The amount of permanganate used will show approximately the Fe plus V present. Add an excess of permanganate and boil at least five minutes. Then add ferrous ammonium sulphate until the MnO_2 is dissolved and the liquid clear, and boil. Cool to about 70° C., add permanganate to a pink color, then run in ferrous ammonium sulphate until a ferricyanide spot shows an immediate blue color. Titrate back with permanganate until the spot shows no blue color in thirty seconds, read the burette and run in permanganate until a pink color is obtained, which persists one minute. The iron value of the permanganate multiplied by .916 gives the vanadium value. If the quantity of vanadium

is so small as to make its presence doubtful it is well to add H_2O_2 to the solution after titrating. This destroys the permanganate color and shows as little as .05 per cent vanadium by the characteristic red-brown tint.

The amount of iron present is approximately shown by deducting the permanganate used in the vanadium titration from the amount required to oxidize the iron and vanadium above.

W and Si are best determined in a separate sample. Treat one gram of the powdered ore with 20 cc. HCl and evaporate to dryness. Add 10 cc. HCl and about 100 cc. water and heat to boiling. Filter and wash by decantation with hot dilute hydrochloric acid until all lead chloride is removed from the precipitate. Add 10 cc. dilute ammonia containing a little ammonium chloride to the beaker containing the WO_3 and silica, allow to stand about five minutes and pour on to the filter, receiving the filtrate into a platinum dish. Wash four or five times using about 2 cc. of the ammonia solution for each wash. Evaporate to dryness, ignite gently until the ammonium salts are volatilized then at a moderate red heat. Do not heat excessively as WO_3 can be slowly volatilized with a strong Bunsen flame. Add a few drops of HF and a drop of sulphuric acid to the dish, evaporate, ignite, and weigh the WO_3.

Determine silica in the residue by loss with HF, and add to this result any silica found with the WO_3.

The Analysis of Ferro-Boron.—After considerable experimenting with various methods suggested for the determination of boron in metallurgical products, the writer has found the following plan to be both accurate and rapid:

Fuse one-half gram of the powdered sample with potassium nitrate and sodium carbonate in a platinum crucible. Pour the melt upon an iron plate, transfer with the crucible to a porcelain beaker and boil with just enough water to effect disintegration. Add a little sodium peroxide during the boiling to precipitate manganese. Filter into an Erlenmeyer flask and wash with hot water. Reserve the residue for manganese determination. Acidify the filtrate with hydrochloric acid and then add a moderate excess of calcium carbonate. Connect the flask with an upright reflux condenser and boil about ten minutes to remove all carbondioxide. Filter, cool to room temperature, add phenophthalein and run in deci-normal sodium hydrate to pink color; then add about one gram mannit. This destroys the color, and more sodium hydrate is added until the color is permanent, even on the addition of more mannit.

One cc. deci-normal sodium hydrate equals .0011 gm. boron. No deduction is required for blank.

One-half gram steel containing 4 per cent manganese, to which .03 gm. boron was added in the form of fused boric acid gave by this method .0299 gm. boron.

Dissolve the residue from the fusion in hydrochloric acid, add 15 cc. sulphuric acid and evaporate to strong sulphuric fumes. Cool, dilute, heat till clear and make up to 250 cc. Take 50 cc. for manganese determination by the bismuthate method, and 50 to 100 cc. for iron determination. Manganese may be made on a separate sample if desired by the bismuthate method without separating the boron.

Silica is determined in the usual way, and carbon by direct ignition in oxygen, spreading the sample over ignited asbestos in the combustion boat.

Primos Chemical Co., Newmire, Col.

The Electric Iron Reduction Plant at Trollhattan, Sweden—II.

In our July issue, page 368, was given a detailed description of the electric iron ore reduction plant at Trollhättan, Sweden, from a paper of Mr. J. A. LEFFLER, presented to the Jernkontoret (The Swedish Association of Iron Masters) at their meeting at Stockholm on May 31. The following is an abstract of the part of Mr. Leffler's paper in regard to the principles employed in the design of the furnace, analysis of materials, operating results, etc.

The Principles of the Reduction Furnace.

The fundamental principles of the construction of this furnace which was invented by Messrs. Grönwall, Lindblad, and Stalhane, is described in a communication from the Electrometals Ltd., of London, England, as follows:

1. As all known materials become good conductors at a high temperature, the construction must be such that the current does not pass away through the lining during the working.

2. The construction must be such that the charge itself protects the lining from destruction through radiation from the electric arc or through conduction; otherwise it would be necessary to employ water-cooling with consequent waste of energy.

3. The charge must not press harder against the electrodes than to permit the formation of an arc. If this condition is not fulfilled the working becomes irregular and the tension becomes unduly low, so that the electrodes and conductors would have to be of inordinately large dimensions.

FIG. 1.—FURNACE STACK.

4. In an electric furnace the carbon charged with the ore cannot burn, and in order to obtain a product of uniform quality it is, therefore, necessary to have a hearth of sufficiently large dimensions, to serve as a mixer. Otherwise every irregularity which inevitably occurs in charging will influence the carbon contents of the product.

In order to fulfil these conditions the hearth is constructed in the shape of a comparatively large crucible covered by an arched roof. When the charge sinks down into the crucible it spreads there in such manner that a free space is always formed between the charge and the brickwork of the roof. The electrodes are introduced into the crucible through the brick work at such points where it is not in contact with the

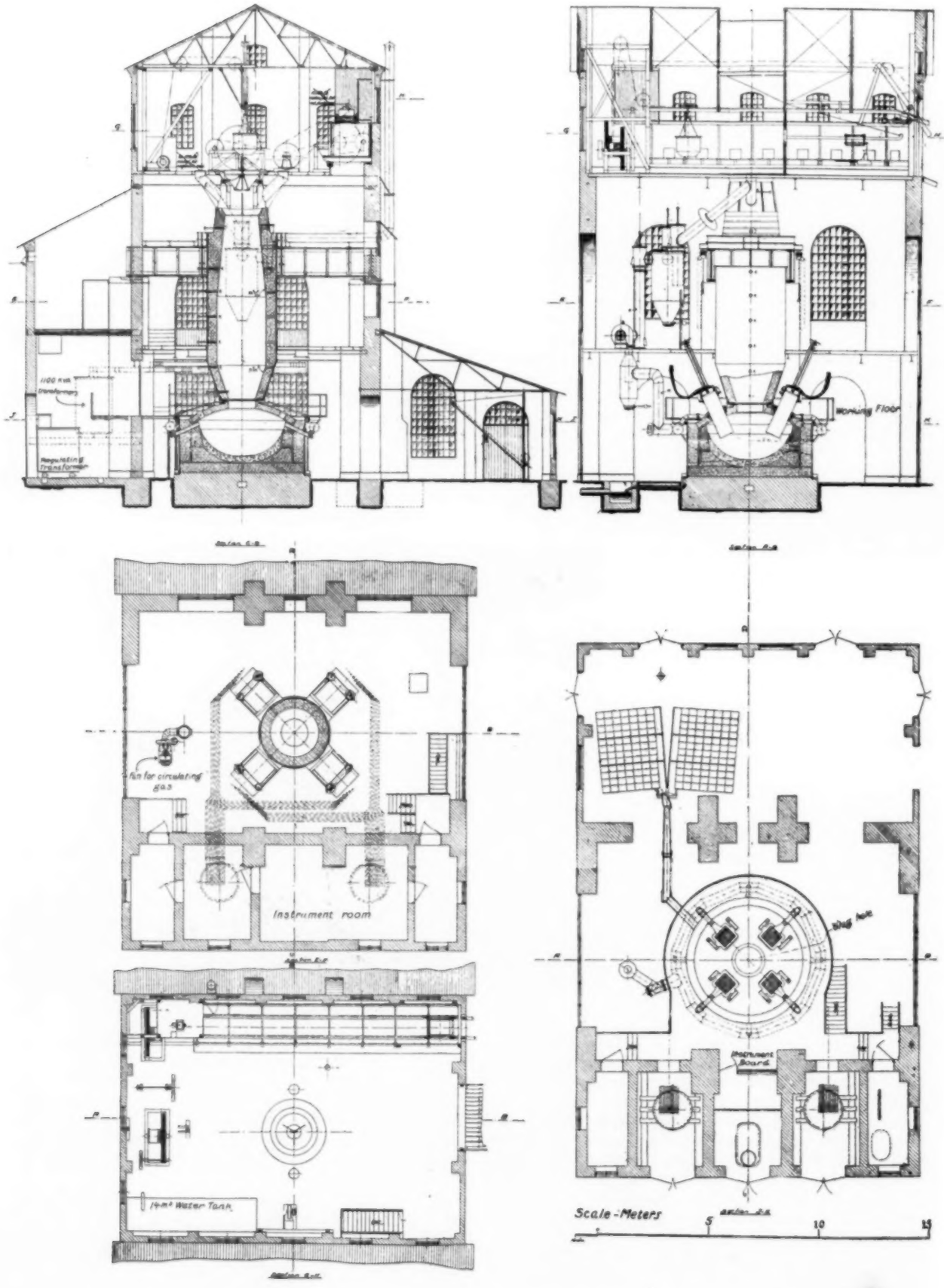


FIG. 2.—DIAGRAM OF ELECTRIC FURNACE CONSTRUCTION AND ARRANGEMENT OF ELECTRIC FURNACE PLANT AT TROLLHÄTTAN.

charge and enter the charge on the free surface exposed. By means of this construction the above enumerated conditions 1, 2 and 3 are fulfilled, and as the hearth can be made comparatively large condition 4 is also fulfilled.

Another important feature in a furnace of this kind is the regulating of the energy supply. In steel refining furnaces,

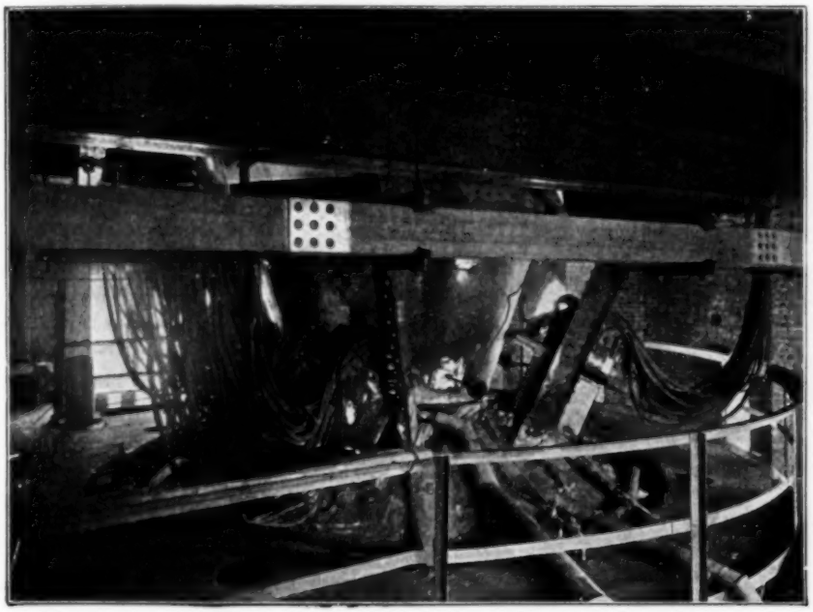


FIG. 3.—ROOF OF CRUCIBLE AND BOSH.

etc., this is done by raising or lowering the electrodes, but as a reduction furnace must necessarily work with a considerable gas pressure, this method cannot be conveniently employed, as it would be too difficult to keep the packings tight around the electrodes. Instead of this a system has been employed by which the electrodes are generally only adjusted corresponding to their wear, but all other regulating is accomplished by varying the tension of the current supply. This is done by cutting out windings on the high-tension side of the transformers and the arrangements are such that the different phases can work simultaneously with different tensions. This method for regulating the current which has been employed for the first time at Trollhättan has proved to greatly facilitate the working.

Principles Employed for Calculating the Form and Dimensions of the Shaft.

The starting point for the calculations was a manufacture of 7,500 tons of pig iron per year of 11 working months; that is about 23 tons per 24 hours. Further the calculations were based on the following assumptions:

1. 1 cu.-meter of charcoal weighs 150 kg.
2. 3 kg. of pig iron is obtained per 1 kg. charcoal.
3. For 1 ton iron 1,725 kg. ore is required, that is, 5,175 kg. per ton of charcoal.
4. One-half of the ore is contained in the interstices between the pieces of charcoal, the other half requires extra space.
5. The specific gravity of the ore is 2.5.
6. The ratio of volume of charge per day to shaft volume 1.55.

It is then found that 23 tons of pig iron correspond to a volume of 51.1 cub. meters of charcoal and an extra space of 7.9 cub. meters for the ore, or altogether 59 cub. meters. As the ratio of the volume of charge per day to the shaft volume is assumed at 1.55 the volume of the furnace should, therefore, be $59 \div 1.55 = 38$ cub. meters.

This consists of the useful volume of the

crucible	12.45	cub. meters
neck	0.45	" "
bosh	3.50	" "
cylindrical portion of the shaft	11.80	" "
stack	9.80	" "
	38.00	cub. meters.

The diameter of the neck was the first dimension to be decided on, and this was fixed at 1200 mm. It appeared desirable to make it larger, but this would have made the crucible still larger and more unwieldy, as the inside diameter would have had to be increased in proportion on account of the electrodes. The depth of the crucible was determined on the strength of the experience with the Domnarfvet furnace and its bottom and sides designed in suitable form.

It is evident that with this diameter of the neck the diameter of the bosh could not be made very large. It also appeared desirable that the height of the shaft should be such that the best possible effect should be obtained from the ascending gas for the preheating and partial reduction of the ore. On the other hand, it might be expected that a too great height of the charge would cause a too great back pressure against the ascending gases. Also the cost of construction would be increased considerably, with an increased height of the shaft.

FIG. 4.—FURNACE TOP.

The problem then was how to best distribute the 25.55 cub.-meters of space that the effective portion of the shaft should contain.

A high and narrow shaft would give a good efficiency from the gas and reduce the risk of scaffolds forming in the bosh.

At the same time no undue increase of the cost of construction should be caused, and an excessive back pressure against ascending gases in the shaft must be avoided. The result of these speculations was the form shown in Fig. 1. (For convenience this is reproduced here again from our July issue.) The reasoning was that if the shaft should prove too high the remedy would be to work with a lower charge. The total height of the furnace was thus fixed at 12.7 m. from the bottom of the hearth to the furnace top. The bottom of the hearth is one m. above the floor level, and the height from the ground level to the furnace top is therefore 13.7 m.

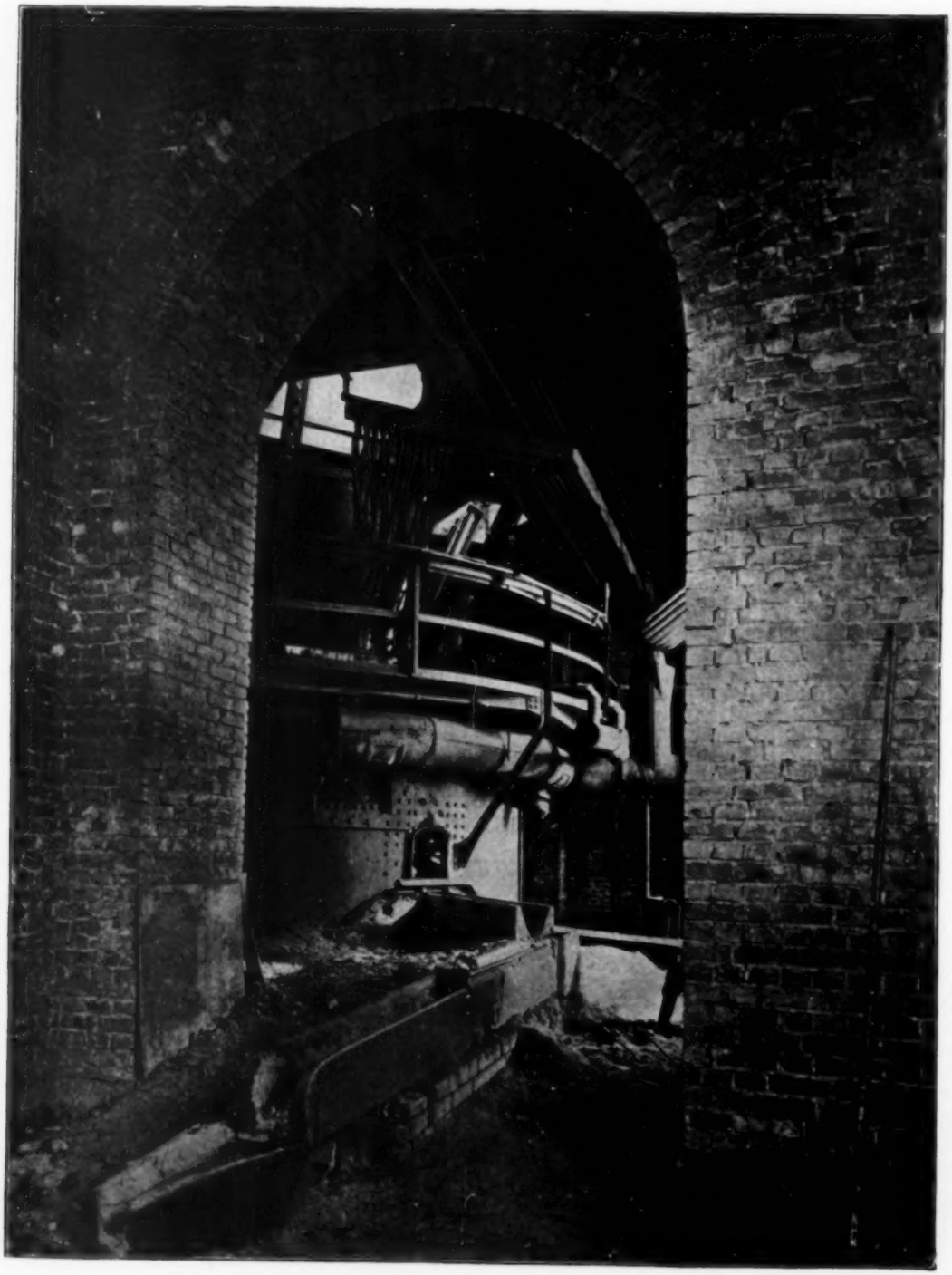


FIG. 5.—FRONT OF FURNACE WITH TAP HOLE.

The Charcoal.

Apart from the small quantity of coke used for starting the furnace, charcoal has been used exclusively for the reduction. The charges are made by volume (0.65 cub. meters per charge), but all of the charcoal is weighed and analysis made every day.

The average composition of the charcoal is 14.58 per cent H_2O, 10.26 gases, 3.06 ash, 72.10 fixed C, 87.95 C, 2.60 H, 0.15 N, 0.61 O.

The average weight has been 168.6 kg. per cub. meter, corresponding to 144.9 kg. per cub. meter of dry charcoal. Both moisture and ash have occasionally been unduly high, in the latter case owing to stones and gravel mixed with the charcoal.

The average results of the daily analysis of the ash give 39.1 per cent SiO_2, 7.3 $K_2O + Na_2O$, 19.55 CaO, 3.85 MgO, 23.1 $Fe_2O_3 + Al_2O_3 + Mn_3O_4$, 0.95 P_2O_5, 3.3 SO_3, 0.6 Cl, 1.0 CO_2.

Besides the regular daily routine analyses a number of special analyses were also made.

The average of 99 special analyses of the ash gave 38.02 SiO_2, 6.1 $K_2O + Na_2O$, 19.52 CaO, 3.59 MgO, 22.69 Fe_2O_3, 3.88 Al_2O_3, 1.89 Mn_3O_4, 1.01 P_2O_5, 0.93 SO_3, 1.35 CO_2.

The analyses have also shown that the volume of gas obtained per gram of charcoal (dry and free from ashes) in in heating to 900 deg. Cent. was 170.35 cu. m. in average. The loss in weight of charcoal (dry and free from ashes) in heating to 900 deg. Cent. was 12.65 per cent. The weight of the gas was 9.22 per cent of the charcoal, and its composition in per cent of weight was as follows:

CO_2	CO	CH_4	H
22.47	38.03	30.47	9.03

Ore and Limestone.

The ore and limestone are broken up in a Blake crusher. This machine is large and powerful, as it was expected that

principally unroasted ore in large lumps would be crushed. Such has also generally been the case.

As the crusher delivered a coarser product than desired, a plate was fitted under one of the jaws so that the ore which at first had been of egg size was reduced to nut size.

The following are a few of the analyses of the limestone and ore used:

Limestone from Gasgrufvan: 54.32 CaO, 0.24 Fe_2O_3, 0.35 MnO, 0.31 MgO, 1.68 SiO_2. Loss on ignition 42.94.

Iron ore from Tuolluvaara: 91.05 Fe_3O_4, 1.41 Fe_2O_3, 0.16 MnO, 1.12 MgO, 0.28 CaO, 0.16 Al_2O_3, 0.10 TiO_2, 3.42 SiO_2, 0.050 P_2O_5, 0.011 S.

Iron ore from Langban: 89.85 Fe_2O_3, 2.49 FeO, 0.18 MnO, 0.85 MgO, 0.99 CaO, 0.42 Al_2O_3, 5.08 SiO_2, 0.014 P_2O_5, 0.007 S.

The chapters of Mr. Leffer's paper, dealing with the behavior of the electrodes and the general working results, will be given in our next issue.

A Plea for an Inventory of the Coal Supplies of the World*

By PROF. JOSEPH W. RICHARDS.

I will take a few minutes of your valuable time to speak on this subject from the standpoint of the metallurgist—the man who uses the ore which the geologist describes and which the mining engineer takes out of the ground.

I cannot agree with some of the detailed conclusions of the very valuable report presented to the Congress upon The Iron Ore Reserves of the World. For instance, the 250,000,000 tons of Lake Superior iron ore carrying between 35 and 50 per cent of iron are placed neither in the actual reserves nor even in the potential reserves and yet this enormous quantity of ore, averaging much better than the "Minette" of Luxembourg, should in my opinion be placed under the actual reserves of ore in the United States. The reason for this statement is that this ore is not now worked simply because we have, at present, all we need of the richer ore, on which we make a larger profit, but if we did not have this richer ore we would certainly be working 40 to 50 per cent ore, and working it at a profit. I believe that within twenty years this immense quantity of ore, which the report does not consider as ore at all, will be the principal supply of iron ore in the United States, and will furnish us ore for many centuries to come.

Aside from the above criticism, I wish to support the points raised by M. DE LAUNAY and Professor KEMP, to the effect that metallurgists are continually improving their methods, and that the improvements which are certain to be made as time passes will enable metallurgists to work poorer and poorer ore at a profit. In twenty years we will be making pig iron from 40 per cent ore as cheaply as we now make it from 50 per cent ore; this is very certain and so I believe that improvement in metallurgical processes will more than compensate for the decreasing richness of the ores.

This very valuable report on iron ores of the world has therefore made it very clear to the metallurgist that there will never, in all the future history of the world, be any scarcity of iron ores for the metallurgist to work upon. Certainly, some localities will become exhausted, and the centers of production may change from place to place, but there will always be enough iron ore in the world to supply its needs.

A much more important question is the quantity of coal in the world with which to reduce this ore. This necessary material will be the real criterion of our future iron supply. Even at the present time the countries which have the most coal are dominating the iron supply of the world.

As already indicated by Prof. KEMP, the International Geological Congress would perform a very great service if it would call upon eminent geologists in all countries to make a careful inventory of the actual and potential coal reserves of the world. Such a report would not be more difficult to make than the present report on the iron ores, but probably less difficult, and it would be a splendid and highly valuable supplement to the report before us, giving us more real information about the future supply of iron and its localities of production. I wish the Congress would take up this task, and give the world this much-needed information.

However, in the distant future, when coal is exhausted, the world must produce iron by the use of the energy of waterfalls and electrical reduction, with charcoal from wood as the reducing agent. Here in Scandinavia, with no coking coal, you have already begun this era, you have been the first to show the world the economic possibility of producing pig iron electrically, using only water-power and charcoal. With your very pure ores, cheap water-power and abundant supply of wood, you will develop in Scandinavia a large electrometallurgical iron industry. It is not a matter of the distant future for you, but within twenty years your iron industry will have been revolutionized and have become electrical. I congratulate Scandinavia on being in the lead in this new era of metallurgy, just as it has always led the world in the production of the purest commercial iron.

*An address delivered before the International Geological Congress in Stockholm, Sweden.

Notes on Chemistry and Metallurgy in Great Britain

(From our London Correspondent.)

The Chemical Society's Faraday Lecture.

The Fundamental Properties of the Elements.

On June 14 Prof. THEODORE WILLIAM RICHARDS, of Harvard, delivered this year's Faraday lecture at the Royal Institution, and was afterwards presented with the Faraday medal. The following is an abstract of the lecture:

The lecture concerns the methods and results of some extended series of investigations designed to add to the foundations of human knowledge by careful experiment. Among quantities worthy of exact measurement the properties of the chemical elements are surely some of the most fundamental because the elements are the vehicles of all the manifold phenomena within the range of our perception. Weight is one of the most significant of these properties, and in the determination of atomic weights each substance employed must be free from all suspicion of impurity. Volume is also important. Twelve years ago the quantity *b* in Van der Waals' equation seemed to the author to be not really constant but subject to change under influence of temperature and pressure. This quantity is supposed to be dependent on the space occupied by the molecules, and if it be changeable are not the molecules themselves compressible?

If changes in the bulk of molecules are to be inferred even from gases, may not expansion and contraction of solids and liquids afford a much better clue to the relative expansion and contraction of these molecules? The ordinary conception of a solid has always seemed to me little short of an absurdity. A gas may be imagined with moving particles far apart; but could a rigid body like steel have such an unstable structure? The evidence seems to show that the interstices between atoms in solids and liquids must be small relatively to the size of the atoms themselves, if, indeed, there are any interstices at all. The compressibilities of 35 elements were studied with sufficient care to leave no doubt as to their relative values. It became manifest that the formation of a compound of a compressible element was attended with greater decrease of volume than was the formation of a similar compound of a less compressible element, other things being equal. This is just what the theory leads us to expect, and is inexplicable by any other hypothesis known to me. Then as to cohesion, if the pressure of chemical affinity causes compression of the atom, may not

the pressure of cohesive affinity do the same? The affinity which prevents solids and liquids from vaporising is generally admitted to set up great internal pressure, and must it not tend to compress the molecules? Molecules of high cohesive affinity—those of substances hard to volatilize—should be much compressed and have a small volume, whilst molecules with little cohesive affinity should be larger. Moreover, those molecules already much compressed by their self-affinity would be only slightly affected by additional pressure. Given two otherwise similar substances, the less volatile would be less compressible, denser, and have greater surface tension. These outcomes of the theory correspond with observed facts in most of the cases so far studied. Differences of structure and of chemical nature sometimes hide these relations, but the bulk of the evidence strongly indicates that cohesiveness, as well as chemical affinity exerts pressure, and that each plays a part in determining the volumes occupied by molecules.

Carried through to its logical conclusion, the idea that atoms are compressible gives one quite a new conception of the molecular mechanics of the universe. The influence of atomic compressibility may be perceived everywhere, and in most cases each fact seems to fit easily and without constraint into its place in the hypothesis. Even apparent exceptions, such as the abnormal bulk of ice, may be ascribed in a reasonable fashion to superimposed effects.

The lecture was received with marked appreciation by a full audience.

The Influence of Temperature on Corrosions.

A paper on "The Action of Salt Solution and of Sea-water on Iron at Various Temperatures," by J. Newton Friend and J. Hallam Brown was read at a recent meeting of the Chemical Society. The authors point out that the results obtained by Heyn and Bauer indicated that solutions of chlorides and sulphates of the alkali metals always exerted less corrosive action on iron than did fresh water at the same temperature. As these results did not conform to the observations of other chemists it was thought desirable to investigate the whole question. Kahlbaum's pure iron foil was used, access of dust was prevented while free passage of air was allowed, unequal illumination was guarded against and equality of conditions was maintained. The loss in weight after 14 days was taken as a measure of the corrosion, and the results showed that at temperatures above 13 deg. C. salt solutions of all concentrations exert a less corrosive action than distilled water, but below this temperature the more dilute solutions are much more corrosive than fresh water. At 13 deg. dilute solutions of salts were practically equally corrosive as fresh water. The results are of importance in this country, where natural waters are usually at a lower temperature than 13 deg. C.

The Measurement of High Temperatures.

At a recent meeting of the London section of the Society of Chemical Industry Mr. C. T. Heycock, F.R.S., read a very interesting paper on the modern methods of measuring high temperatures. The discussion was opened by Mr. Darling, who said he had the impression that Siemens had given up his platinum resistance pyrometer, and that nothing further was effected with it until Callendar and Griffiths worked on it; but the difficulty arising from the heating of the stem was overcome by covering the platinum wire with magnesia instead of winding it on mica. The relative merits of resistance pyrometers and thermo-couples depended on the use to which they were put. For exact determinations of boiling points or melting points the platinum resistance pyrometer gave the closer results; but if the two instruments were compared in a furnace at a high temperature for a considerable time experience showed that for workshop purposes the thermo-electric was the better. Prolonged heating of platinum wires at slightly over 1000 deg. C. materially altered their resistance, and the calibration was then wrong. Taking a reading even with such indicators as the Whipple was a troublesome matter, involving balancing with the Wheatstone bridge, and, even in skilled hands, by the time a balance was obtained the temperature might have varied considerably.

With the thermo-couple a direct reading was got at the instant, so that it had a decided practical superiority. The deteriorations in thermo-junctions occurred equally with platinum, and consequently the thermo-electric instrument was not inferior in this respect. As to accuracy, it was not possible with a large furnace—gas or coal-fired—under workshop conditions to maintain a constant temperature within ten degrees, and it was useless to employ a method giving readings to a fraction of a degree. There were many instances—notably the Royal Arsenal at Woolwich—where after thorough trial resistance pyrometers had been given up and replaced with thermo-couples, mainly because an installation of the latter could be worked from a central control if all the wires were identical in composition.

It was only fair to say that resistance pyrometers had recently been made quite automatic, but it was not possible to adapt even such an instrument as the new Paul indicator to an installation of 50 resistance pyrometers. Another important advantage of the thermo-couple was the facility of repair, while if the resistance instrument got out of order it was best to return it to the maker at once. For laboratory use in getting precise results no doubt the resistance pyrometer was the better.

He wished to refer to a curious effect observed with the Féry pyrometer in which a spiral was employed. When heat was concentrated on that spiral the pointer moved to a certain point on the scale, remained stationary for a moment or two, and then moved further. Which reading should be taken? They recommended taking the first, but that was not always correct, and there was something about it which required investigation. The form with the spiral was not so nearly accurate as that with the thermo-junction and galvanometer.

As a practical example of the accuracy now attainable at high temperatures he would state that in taking temperatures at about 1500 deg. C. some time ago with a Féry pyrometer and an Holborn-Kurlbaum optical instrument, by Siemens, as a check, the two agreed perfectly and responded equally to fluctuations of about 10 deg. either way. They need not fear that pyrometers could only be used in the laboratory, for they could be used with confidence in the workshop.

Dr. R. Lessing asked how readings on the Féry pyrometer were affected by glass or mica windows. He found it best to put the glass on for adjusting the distance and remove it for the actual reading, which was, of course, higher than the reading through the window. He wished that the author had touched on the subject of testing by Seger cones. It was not a very exact method, but was used largely in ceramic industries and had been brought to some degree of perfection; moreover, it could be employed where the use of pyrometers was inconvenient or even impossible.

Prof. W. R. Hodgkinson thought Mr. Heycock had rather neglected the thermo-couple. He considered that it was the pyrometer of the day on account of its simplicity and permitting the attachment of a recorder. In metallurgical works where pyrometers were in constant use, night and day, a recorder was wanted to show what had been going on at any time. True, there was the fact that Rhodium was attacked by carbon compounds, carbon monoxide, etc., and therefore must be protected. No doubt some vapor—probably a carbon compound—came from the iron, but when the wires were enclosed in a silica tube difficulty from that source was overcome.

Mr. W. Macnab said he had no experience of resistance pyrometers, bue he had found the thermo-couple a most convenient instrument for the practical determination of high temperatures.

Mr. Gardner asked if it were quite certain that iridium volatilized.

Professor Hodgkinson: There is not the slightest doubt about that.

Mr. M. J. Butterfield remarked that Mr. Heycock had stated that the range of nitrogen-filled glass thermometers was below 400 deg. C., but they were now made of "boro-silicate" glass with scales extended to 550 deg. and 600 deg. C. They were very convenient, and he found them in fair agreement with thermo-couples.

Mr. Heycock, replying, said his object had been to point out the value of the resistance pyrometer for exact work, and he quite agreed that for practical workshop use it was not so convenient as the thermo-couple. He should hesitate to use a glass or mica window, but would employ a plate of rock salt. He was interested by what had been said about Seger cones, and confirmed Dr. Lessing's observations. As to the volatilization of iridium, the resistance of an alloy of iridium and platinum at high temperatures underwent a much greater change when kept hot for some hours than that of ordinary platinum.

Heat Treatment of Steel.

A. JUNG has published some interesting details of his researches on the influence of heat treatment on the tenacity and microstructure of hyper-eutectoid steels contained from 0.99 to 1.56 per cent of carbon, 0.24 to 0.46 per cent of manganese, 0.019 to 0.032 per cent of silicon, 0.020 to 0.031 per cent of phosphorus and no sulphur. These were heated in an appropriate carbon resistance furnace, and the changes in the values obtained by mechanical tests were noted as well as the microstructure after the following variations of treatment: (a) Heating for equal times at different temperatures and hardening in water; (b) tempering at the temperature found to give the best results and noting the results obtained with variation of time; (c) quenching in baths of oil and lead-tin alloy at the same temperature as the tempering was effected at; (d) tempering of the steels hardened in water both at varying temperatures for equal times and for varying times at the same temperature; (e) annealing and subsequently noting the results obtained on mechanical tests, contraction of area, nature of fracture, quality of grain and hardness.

The investigator has arrived at the following conclusions: Such steels are best hardened at temperatures confined to a narrow range, which is as the carbon content; for any given carbon content the ultimate strength is inversely as the temperature at which quenching is effected; at constant quenching temperature the ultimate strength is as the carbon content; the quenching temperature should be the same for all baths; quenching in water and tempering at 350° C. produces the greatest degree of strength; by quenching in water at a low temperature and tempering a somewhat brittle but highly tenacious metal is obtained, but when quenching is done at the same temperature as the tempering the tenacity becomes less while the toughness increases; identical results cannot be obtained from varying kinds of heat treatment, and can only ensue with any one property of the steel; the mechanical properties are dependent on the conditions in which the carbon exists and on the metallic structure; structures containing martensite give the highest degree of hardness, tenacity is increased by troostite and sorbite as elongation is by pearlite; finally no consistent influence of the carbon content on the mechanical properties of the steels was detected.

The Cementation of Nickel Steel.

An investigation by F. CARNEVALI and F. GIOLITTI on steels containing from 2.03 per cent to 29.8 per cent of nickel, when cementation was effected with pure ethylene and carbon monoxide, shows that the maximum carbon content in the zone of cementation diminishes as the percentage of nickel increases, and becomes very marked when there is more than 5 per cent of nickel. The variation of concentration of carbon in the layers of the zone of cementation is more regular in nickel steel than in carbon steel. When nickel steels are so cemented as to show a hyper-eutectic zone, the phenomena occurring during slow cooling are similar to those exhibited by similarly treated carbon steel, but differ quantitatively from the latter in that the entity of the liquation phenomena is not so marked in the nickel steels, and the transition from the hyper-eutectic to the hypo-eutectic zone corresponds with that of a carbon steel having a lower carbon value, viz., 0.6 to 0.65 per cent as against 0.9 per cent.

Electrolysis of Sodium Chloride Solution.

R. SACERDOTI has investigated the effect of rise of temperature on the potentials of the electrodes in the electrolysis of sodium chloride solution, with platinum and with iron cathode and platinum and magnetite anodes. In the cathode curve the effect of temperature is but slight, but in the anode curves it is apparent that as the result of a rise of temperature from 20° C. to 100° C. the anode potential corresponding to a given current density falls 0.4 to 0.8 volt.

A diagram shows the relation between electric potential and temperature for different current densities at both electrodes, and demonstrates that the reduction in voltage due to rise of temperature is never more than 0.75 volt, and that when high densities are used the figure is always remote from the potential of reversible polarization. With a decidedly alkaline solution the reversible anode potential is 1.3 volts and the reversible cathode potential 0.8 volt.

Copper, Nickel, and Cobalt Ores.

An investigation by H. PEDERSEN of a mixture of magnetic pyrites, nickel ore and slag consisted in digesting the mixture with concentrated nitric acid and treating with sulphuric acid until SO_3 was evolved; when the solution was cold it was treated with water and again evaporated until fuming occurred. The residue was treated with water, and after filtration from the slag the filtrate was precipitated with ammonia, and the ferric hydrate dissolved in dilute sulphuric acid, and after reprecipitation was estimated by the usual titration. The solution containing the copper and cobalt was acidified with sulphuric acid, evaporated to a convenient bulk, and after addition of a small quantity of strong sulphuric acid the copper was estimated electrolytically. The solution containing the nickel and cobalt was neutralized with ammonia in distinct excess and the solution submitted to electrolysis. The nickel and cobalt on the cathode were dissolved in excess of nitric acid, the solution evaporated to dryness on the water bath. The nitrates were eliminated by a small addition of hydrochloric acid and repeated evaporation, then redissolved in dilute hydrochloric acid and suitably diluted; the nickel was then precipitated by an alcholic solution of dimethylgloxime and ammonia; the precipitate was collected in a dried and weighed Gooch crucible, and after drying for about two hours at about 110° C. the precipitate contains 20.31 per cent of nickel.

Engineering Imports and Exports.

According to the Board of Trade returns for the six months ended with June, the imports of iron and steel, including manufactured articles, had a value of £5,685,566, an increase of £1,343,165 on the value for the corresponding period of 1910; the exports amounted to £22,160,641, and showed an increase of £1,212,451. For other metals the imports were £13,446,938, an increase of £1,854,921, and exports were £5,485,763, an augmentation of £443,020. In electrical goods there is a decrease of £32,307 in imports and also a decrease of £338,361 in exports. The value of imported machinery is £3,148,648, an improvement of £867,086; while exports amounted to £15,729,408, showing an increase of £1,648,922. The value of imported ships is £47,940 and of exported ships £2,532,246, and there are respectively an increase of £45,950 and a decrease of £1,768,181.

Market Prices.

July, 1911.

Copper has been fairly steady throughout the month, with

downward tendency, dropping more sharply after the 24th. Opened at £57-5-0 and closes at £56-7-6.

Tin rose over the first fortnight, though not steadily, opening at £194-10-0 and reaching £196-15-0 by the 14th. It then dropped sharply away to £188-10-0, recovered to £193-10-0 by the 19th, and closes £189.

Lead was steady at £13-15-0 to the 18th, when it commenced to rise, reaching £14-3-9 by the 26th. Closes at £14-5-0.

Hæmatite dropped after opening at 62/- to 60/10, recovered by the 10th, and has since been from 61/3 to 61/4½.

Scotch Pig rose from 52/3 smartly to 53/-, but has since eased to 52/10.

Cleveland rose from 46/3 to 47/- and remained fairly steady, closing at 46/10½.

Rubber rose but has not exhibited very extensive fluctuations, being from 4/1 to 4/7 (19th) and closing at 4/8.

	£ s. d.
Alum, lump, loose, per ton	£ 5 12 6
Antimony, black sulphide powder, per ton	22 0 0
Borax, British refined crystal, per ton	16 0 0
Copper ore, 10 to 25 per cent, per unit	98. to 10 0
Copper sulphate, per ton	19 13 0
Carbolic acid, liquid 97/99 per cent, per gal	1 4
Creosote, ordinary good liquid, per gal	2¼
Camphor, 1-oz. tablets	1 9½
Caustic soda, ash 48 per cent, ordinary, per ton	5 10 0
Litharge, flake, per ton	15 15 0
Mica, small slab, original cases, per lb	6d. to 2 0
Naphtha solvent, 90 per cent to 160° C.,, per gal	9
Petroleum, Russian spot	5½
Sal ammoniac, lump, firsts, delivered U. K., per ton	42 0 0
Sulphur, recovered, per ton	5 0 0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14 0 0
Shellac, standard T. N., orange spots, per cwt	4 4 0
Tinore, 70 per cent, per ton	£123 to 125 0 0
Hydrochloric acid, per cwt	5 0
Zinc, Vielle Montague, f.o.b. Antwerp	30 12 6
Quicksilver, per bottle	9 0 0
Platinum, per oz	8 12 6

The following have risen since end of June, 1911:

Lead	£ 10 0
Hæmatite	4½
Scotch pig	7
Cleveland	7½
Rubber	7
Litharge	5 0
Petroleum	1⅛
Sulphate of ammonia	12 6
Quicksilver	12 6
Zinc V. M.	17 6

The following show a falling off in price:

Copper	£ 7 6
Tin	5 10 0
Copper sulphate	3 7 0
Carbolic acid	1
Tin ore	3 0 0

The stamp-tube mill combination for fine grinding, so common in South Africa, is being assailed as inefficient from a mechanical standpoint. Many engineers are inclining to the belief that other machines, particularly rolls and Chilean mills, will do the work with greater mechanical efficiency.

The principal use of fuller's earth in this country is in bleaching, clarifying or filtering fats, greases and oils. The common practice with mineral oils is to dry the earth carefully after it has been ground, and run it into long cylinders through which the crude black mineral oils are allowed to percolate very slowly. As a result the oil which comes out first is perfectly water white and much thinner than that which follows. The oil is allowed to continue percolating until maximum shade is reached. The earth is then purified.

SYNOPSIS.

Gold and Silver.

Cyaniding Antimonial Gold Ore.—An interesting experience in the treatment of a gold ore carrying some stibnite is detailed in the July, 1911, issue of the *Mining Magazine* (London), by Mr. H. T. Brett.

The Globe and Phoenix mine in Rhodesia produces a hard white quartz ore almost free from base minerals except for the presence of antimony as sulphide. This mineral occurs in average quantities of about 1 per cent, but in some places as much as 3 per cent is found. An analysis shows: SiO_2, 76.32 per cent; Sb, 0.73 per cent; Fe, 2.58; S, 0.91; CaO, 1.21; MgO, 6.44; As, Trace; Cu, Trace; Au, 1.6 oz. per ton; Ag. 6 oz. per ton.

More or less difficulty has been experienced in the treatment of this ore on account of the effect of the antimony. For example, in cynaiding sand and slime, with everything favorable for good extraction, the presence of only 1 per cent of stibnite makes an extraction of more than 20 per cent of the gold practically hopeless. Concentration is difficult on account of the friability of the antimony mineral.

The original plant erected for the treatment of the Globe and Phoenix ore consisted of 40 stamps, a sand-leaching plant and a slime decantation plant. When the antimony appeared in the ore, the extraction dropped to 30 per cent on slime and 40 per cent on sand. It was then discovered that by allowing the slime to weather for six months it was more amenable to cyanidation, and this course was then adopted. But the sand did not yield to such treatment and, moreover, rusty gold began to appear in the ore, thus complicating matters. Various attempts were made to overcome difficulties by regrinding the sand in pans, and by sorting out as much antimony as possible on sorting belts and jigs. The latter proved disappointing. All roasting of the ore gave satisfactory results, but the installation of a plant for this process would disturb existing conditions too seriously. Finally after much experimenting on a large scale a method of treatment was adopted which includes various features which were found valuable.

The ore is now crushed and washed in a trommel 10 ft. 6 in. long by 5 ft. 6 in. diameter at the large and 2 ft. 9 in. at the small end, perforated with 1½-in. holes for the first 4 ft. of its length and ¼-in. holes for the remainder. The trommel oversize is delivered onto a sorting belt from which massive stibnite is sorted in quantities ranging from ½ ton to 1 ton per day. After sorting, the oversize of the trommel is combined with the undersize and delivered to the mill bins.

Forty stamps of 1250 lb. weight crush the ore through 25-mesh screen. It then flows over amalgamated plates, the lower 3 ft. of which are covered with blankets to catch coarse rusty gold. This product is afterward amalgamated in a barrel. The tailing leaving the plates is classified into sand and slime. The sand is concentrated on vanners to remove as much pyrite and stibnite as possible. The concentrate is sent to the roasting plant while the sand tailing is reground in grinding and amalgamating pans and combined with the classifier slime. This fine pulp is then run over blankets where a fine concentrate is recovered, which is later treated in the roasting plant together with the coarse concentrate from the vanners. The fine tailing then flows to the sump of the tailing dump, whence it is pumped to a large spitzkasten for the separation of fine sand and slime.

The fine sand is collected for leaching in tanks which are filled to a depth of 6 ft. Each charge is subjected to from six to eight days actual treatment, but even then the extraction is unsatisfactory, and on ore carrying 1.5 oz. gold the tailings are too rich to throw away. It has been found that as soon as antimony on the sand charge exceeds 0.2 per cent the extraction drops rapidly.

The fine slime from the last spitzkasten is dewatered and

pumped to the storage dams. The clear water recovered from the slime is returned to the stamp mill. As soon as a dam accumulation has reached a depth of 6 ft. and contains about 10,000 tons dry slime, the stream is turned to another dam, and so on. After two months' time the dry accumulation is plowed and harrowed, and finally when the surface appears to be dry and thoroughly oxidized it is taken to the plant, mixed with cyanide solution in proportions of about 1½ parts solution to 1 of slime. This treatment of the surface of the dam is repeated until all the accumulated slime has been removed. Cyanidation of the slime is carried on in the decantation plant originally erected for this purpose, but a vacuum filter plant is under consideration. The extraction of the weathered slime is 75 to 80 per cent. The same material unweathered gives an extraction of only 30 per cent.

The pyrite and stibnite concentrates mentioned above are roasted in a Merton roaster, ground and amalgamated in pans, in which cyanide solution is used. It then flows over blankets where rusty gold is caught. Finally the pulp is agitated for 24 hours and filtered in a Dehne press. The residue is taken to the dump for oxidation and future treatment if required. The raw concentrate contains about 5 per cent Sb and 9 per cent S, while the roasted material carries about 3 per cent Sb and 0.72 per cent S, thus showing the difficulty of eliminating the Sb. The extraction varies from 85 to 90 per cent on a head value of from 2 to 15 oz. per ton. The following table shows the percentage recovery from the various materials treated:

Source.	Value per ton milled.	Per cent per ton milled.
Stamps and pans	$28.94	79.86
Concentrate	2.41	6.65
Sand	0.39	1.08
Slime	2.15	5.91
	$33.89	93.50

Slime Settlement.—Metallurgically speaking, the term slime applies to the finest portion of crushed ore, while sand is the coarsest. Various definitions have been proposed, but the foregoing statement is believed to cover the essentials as far as mill practice is concerned. The settling of slime is an important matter in the present practice of cyanidation, and various means have been adopted for accomplishing this end. In the *Monthly Journal* of the Chamber of Mines of Western Australia for May, 1911, Mr. B. L. GARDINER writes regarding the method adopted at the Sons of Gwalia mine.

The plant was originally equipped with nests of spitzkasten, but the settlement was inefficient and some means was sought of improving the efficiency by materially altering the apparatus then in use. The expedient hit upon resulted in practically increasing the settling area, and in delivering a thicker pulp than before. It consists simply of suspending in the pulp in an inclined position a number of sheets of corrugated iron. The idea is not altogether new, as it has been adopted in the United States and is one of the features embodied in the thickener of John E. Rothwell, made by the Colorado Iron Works Company.

The idea may be illustrated by filling a glass cylinder with slime and then noting the different rates of settlement when the cylinder is placed in a vertical position and at an angle of 45 degrees from the vertical. In experiments made by the author the rate in the vertical cylinder was 40 mm. per minute, while the rate in the cylinder set at an angle was 180 mm. per minute, or four and one-half times as great. The reason for this is obvious on close examination, for, on the upper side of the inclined cylinder a stream of clear water can be seen rising, while the particles of slime settle in a vertical direction from the inclined side of the cylinder.

The effect of the inclined sheets of corrugated iron in the spitzkasten is to reproduce many times the effect noted in the inclined glass cylinder. Wherever there is a surface facing downward the pulp must settle away from it, leaving the water to be forced up. The increased settling area obtained by this arrangement is stated by the author to be the total area of the sheets by the cosine of the angle they make with the vertical, or the projection of their area to the horizontal plane.

To test the efficiency of the inclined sheets, one section of spitzkasten was fitted with them, while three sections were allowed to remain in their original condition. The section fitted with the accelerating sheets showed a fall of slime at the rate of 0.168 in. per minute, while the three unaltered sections gave a fall of 0.107 in. per minute. The altered section gave a thickened pulp containing 41.1 per cent dry slime, while the three unaltered sections showed only 26.1 per cent.

The principle has its limitations, but a simple test with the glass cylinder will show what may be expected from any given pulp. The advantage of the device lies in the fact that it can be applied to existing spitzkasten.

Assay of Silver-Bearing Ores.—To test the effect of varying quantities of different reagents on the assay of silver ores CHARLES R. KEYES and D. F. RIDDELL made many experiments which are recorded in a paper published in the July, 1911, *Bulletin* of the A. I. M. E.

The ore analyzed as follows: Fe, 8.7 per cent; Pb, trace; Cu, 25.6; Zn, 6.2; Mn, 1.0; CaO, 15.0; S, 10. Assays were made to determine (1) how acidic a charge could be run; (2) how basic a charge could be run; (3) how large an amount of borax could be used in a charge; (4) how the normal charge for the kind of ore would behave, and (5) how the use of an excess of litharge would affect the results.

The conclusions deduced from the data given are: 1. A dead roast is not so satisfactory nor so accurate as a run with a large excess of niter or a scorification on a charge of 0.1 A. T. or more. 2. An excess of borax glass gives low silver results, especially in the presence of large amounts of sulphur, copper, iron or other matte-forming substances. The use of amounts approaching 1 A. T. must give low results. 3. An excessively acid flux fuses with great difficulty and yields poor buttons. In the presence of matte-forming substances the silver determination is low. 4. A very basic flux is open to all the objections that are raised to a very acid flux. Both give variant results most of the time. 5. Ores high in copper cannot be run by any crucible method without scorifying the lead buttons. 6. Direct scorification on high-grade copper ores does not check as well as the combination of crucible and scorification. 7. The normal or usual charge for coppery ores gives as high results as any method on this type of ore, but the checking is not as good as when an excess of litharge is used. 8. For ores high in copper, or carrying some zinc, a large excess of litharge in the flux improves the buttons and renders them easily handled. Furthermore, the time and temperature of fusion are decreased, and the buttons check well, with very close to actual valuations for the silver in the ore.

Cyanidation in Korea.—That improvement is the order of the day in all parts of the metallurgical world will be seen from the following description of present and proposed practice at the Unsan mines, Korea. The notes are abstracted from an article by G. C. EVANS in the June, 1911, issue of the *Mining & Engineering Review* (Australia).

The company has five mills, in all of which the same practice is followed. The ore is crushed by stamps, roughly classified in V-boxes for the purpose of getting some coarse sand to mix with the concentrate prior to cyaniding. The entire pulp with the exception of the sand mentioned is then concentrated on vanners. The vanner tailings run over canvas tables and then to waste. The concentrate is given an 18-day treatment in leaching tanks. Under this system the recovery is about 85 per cent. The vanner tailings average about 70 cents per ton and the recovery is 86 per cent; the concentrates are valued at about $22.30 per ton and the recovery is 84 per cent. The milling costs, exclusive of cyaniding, are 54 cents per ton, and the cyaniding costs $1.35.

A modern cyaniding plant is to be installed to treat the con-

centrate, together with an accumulation of old tailings amounting to 195,507 tons. The old tailings are valued at about $4.75 per ton. The new system will include concentration on Wilfley tables, grinding of the table concentrate in tube mills, agitation in Brown tanks, filtration in Dehne presses, and precipitation of the gold bearing solution in Merrill presses with zinc dust. This plan, it will be seen, embodies most of the latest ideas in cyanidation, and is expected to prove profitable to the company.

Mercury.

Wet Process for Cinnabar.—Little work has been done in improving methods of treatment of mercury ores, except in the line of furnaces for volatiling the metal. Low-grade ores have hitherto been without a suitable method of treatment. Mr. C. A. Mulholland has devised a wet process which he believes is practicable in view of the improvements made in the handling of large quantities of wet material. The details of the process are given in the *Aust. Min. Standard* for June 8, 1911.

The process consists in dissolving from the ore the very finely divided cinnabar, and reprecipitating it in a concentrated form ready for distillation in a comparatively small apparatus, in which efficient condensation can be carried out, and the escape of fumes prevented. It has long been known that cinnabar is soluble in alkaline sulphides containing free alkaline hydrates, and may be reprecipitated by the addition of suitable chemicals or electrolized for the recovery of the metal.

According to the author's experiments, the ore could be crushed to ¼ in., and then slimed with the solvent in a tube mill, or other convenient machine. Agitators could be used following the tube, if found necessary to complete solution, and these agitators should be heated by a steam jet. The solvent solution should contain at least one molecule each of alkaline sulphide and hydrate to each molecule of mercuric sulphide, but in practice an excess would be used. When solution is complete the pulp is settled and the clear solution decanted for precipitation. The residue is washed with weak solution and the washings used for extracting fresh ore.

The pregnant solution is now precipitated with zinc hydrate, when zinc and mercury sulphides will fall out of solution and are separated by filter pressing. The mixed precipitate is treated with dilute sulphuric acid and steam which will dissolve the zinc as sulphate, leaving a mixture of mercury sulphide and mercury. This is mixed with lime and iron and retorted in the usual way.

In treating the mixed zinc and mercury precipitate with acid, sulphuretted hydrogen is evolved, which is passed into the filter press effluent to regenerate alkaline sulphide for further use.

From small scale experiments the author has calculated the probable cost of operation on the basis of treating 100-ton lots of ore of varying grades, from 0.05 per cent to 1.0 per cent mercury. For example, on ore of 0.3 per cent grade, worth about $4 per ton, he believes that an efficiency of 80 per cent could be obtained, or a value of $3.20 per ton, at a cost of about $1.85.

Lead and Copper.

Concentrating Table.—In the July, 1911, *Bulletin* of the A. I. M. E. Mr. S. A. Krom presents some considerations on the proper form of table best adapted to the separation of minerals from gangue by methods depending on difference of specific gravity. He rather deprecates the use of riffles, and even states that in the near future they may be entirely eliminated from tables. He believes that riffles rather upset and retard the process of stratification which they are supposed to promote. He argues further that no concentrate can be delivered from a riffled table until the deck has become bedded, and that this bed must be maintained by very carefully adjusted feed of pulp and rate of discharge, or else the efficiency of the table is lowered. Perhaps Mr. Krom is too severe on the riffled table, and as he is engaged mainly in making out a case for an unriffled table, his criticisms will be taken properly modified by that fact.

The author describes a table invented by U. S. James, in which certain radical departures have been taken from the ordinary table. Stratification is obtained in a settling basin into which the table feed runs, and which is formed by two planes of different inclinations to each other, meeting in a line diagonal to the motion of the table. From this basin the concentrate is discharged over the deck of the table by reason of the table motion, and the tailings flow to the discharge. It is claimed that the settling basin does not require bedding and that it is difficult to overfeed. Adjustments in the eccentric motion of the table are made by an adjustable pin, which gives a wide range of different movements.

Copper Smelting in Michigan.—The simplicity of operations in smelting concentrates of native copper in Michigan has given rise to the idea that there is no metallurgy in Lake Superior smelting. In the School of Mines *Quarterly* of Columbia University for July, 1911, Mr. Henry D. Conant gives a descriptive statement of smelting operations in Michigan in which he shows that despite the simplicity of the operation there is opportunity for the engineer to show his ability in the effective use of fuel, the efficiency of labor, the production of clean slags, and the proper refining of the impure metal.

Lake Superior copper is obtained by concentrating the conglomerate or amygdaloid rock, which contains from 1 to 3 per cent copper, and obtaining what is locally called "mineral," containing 60 to 80 per cent copper. This concentrate is smelted in reverberatory furnaces without any fluxes, the slag skimmed off and further treated in cupola furnaces, and the reverberatory copper refined in either the same furnace or a second reverberatory into which it is tapped. In the first smelting operation the impurities are largely oxidized and slagged off. Some of the copper is at the same time oxidized, forming copper suboxide, which must be reduced again. This is accomplished by poling with green wood, after which the copper is cast into forms suitable for the various needs of commerce. Both the oxidizing and reducing operations are governed by the experience of skilled workmen, who take samples from time to time, and determine the end of the reaction by the appearance of the sample.

The reverberatory slags are either acid or basic, according to whether they are produced from amygdaloid or conglomerate mineral respectively. These are properly fluxed with lime, iron and silica, and reduced in cupolas, under a blast of from 3 to 4 oz.

The author describes in a general way the four plants now operating in Michigan, and gives some interesting historical data on the changes that have occurred in practice in later years.

Electrochemistry.

Fixation of Atmospheric Nitrogen with Intermittent Load.—The problem of utilizing the electrical energy of central stations at times of low load has always been attractive for electrochemists on account of low power costs, while electrochemical processes have been attractive for central station engineers to equalize their load curve. The question whether electrochemical processes can be operated intermittently with a financial profit has therefore often been discussed, but there is very little exact information available from commercial working. A note in a recent number of the London *Times* Engineering Supplement is therefore of considerable interest. As abstracted in London *Electrician* of August 4, it states that "for some time past experiments have been in progress at Legnano, Italy, with the object of utilizing the excess power generated during the night and certain hours of the day by a large hydro-electric station. The aim was to produce nitric acid and nitrates from the nitrogen of the air, and it is considered that the problem has now been satisfactorily solved. Some months ago the employment of 4000 kw. for the production of nitric acid was initiated, and now preparations are being made for the use of 9000 kw. more, energy being supplied by the Societa Lom-

barda per Distribuzione dell' Energia Elettrica at a pressure of 50,000 volts."

Electrolysis Destruction of Water and Gas Pipes Due to the Stray Currents from the Rails of Tramway Systems.—Considerable trouble was experienced at Pittsburg, Pa., with stray currents from the rails of the tramway system attacking underground structures of the water system. A very full discussion of these troubles is given in a paper by E. E. Lanpher and L. B. Smith before the Engineers' Society of Western Pennsylvania and published in the *Proceedings* of the Society for July. The authors conclude that the only certain method of preventing electrolysis is the installation of the double-trolley system and the most successful method of mitigating electrolysis is the installation of the "return pipe feeder system." The latter system was introduced in Pittsburg, but this solution was criticised in the discussion by various speakers especially by Prof. A. F. Ganz who favors the radial insulated return feeder system; the return feeders being designed to drain the rails of current at many selected points, leaving these points at the same potential, thus greatly reducing the rail drops and minimizing stray currents on underground structures. In the reply of the authors the unusual conditions prevailing in Pittsburg are pointed out but they do not claim that under different conditions the results would have been equally satisfactory, in fact the conditions as they existed in Pittsburg were almost ideal for the installation of the return pipe feeder system.

With respect to **local electrolysis** brought about by impurities in steel pipes some interesting remarks were made in the discussion by Mr. F. N. Speller of the National Tube Company. To correct a misapprehension that is very widespread as to corrosion being often due to irregularity in the composition, he said that they made a great many experiments along that line which indicated that steel or iron has to be much below the average before the irregularity incident to commercial material became an important influence in corrosion. As a matter of fact, the difference in potential between the greatest irregularity which is ordinarily found due to segregation does not begin to compare with the differences due to the accidental presence of foreign material such as mill scale on the surface of the metal itself. If all the scale could be removed from the pipe, one would probably have much less trouble with the corrosion of underground piping.

Primary Battery.—In our August issue, page 401, a paper by Mr. W. R. COOPER on the Benkö cell was given. In connection with this it is interesting to note a recent exhibition by Mr. Cooper of the Benkö cell at a meeting of the Physical Society in London. Several types of this cell were exhibited. The battery depends essentially upon the use of a porous carbon electrode in the form of a pot (open at the top) which is surrounded by a closed lead shell. The elecrolyte (a bichromate solution) is fed into this shell under a suitable head to give a flow of ½ liter or more per hour per cell. It passes through the carbon to the zinc, where it is removed by an overflow pipe. Polarization is thus reduced to a minimum and remains constant for a given current, provided the cell is not overloaded. The size of cell referred to is 9 ¼ in. high overall, 6 ¼ in. long and 1 ¾ in. wide; it weighs about 10 lb., excluding the tank for electrolyte, and will give a steady current (depending upon the strength of the electrolyte and rate of flow) up to, say, 25 amps. at 1.5 volts. The internal resistance is below 0.01 ohm. There is no "mess" and no difficulty in keeping the apparatus in order. The cell is therefore very suitable for laboratory use or work where the cost is not of prime importance. Various types are being made according to the work required, and these include cells which depend for their action upon diffusion as distinct from a steady flow.

In the discussion, which followed, G. L. Addenbrooke asked whether the contact between the lead and the carbon remained satisfactory over a long time. The author replied in the affirmative so far as his experience (extending over 15 months) had gone. Prof. Callendar inquired as to the maximum number of watts that could be taken from the battery continuously, and whether, weight for weight, more power could be taken from it than from a lead storage battery. The author replying to the first part of the question said about 25 to 40 watts, depending on the solution, and stated that, excluding the reservoir, 2.4 to 4 watts per pound could be taken from the battery as compared with ½ watt (8-hour rate) up to 2½ watts (automobile cells at 5-hour rate) for lead cells.

Mr. W. Duddell asked how the battery would compare with lead cells for giving a steady current of the order of 5 amperes to 1 part in 5,000 or so. Lead cells could only be relied on to keep steady to about one part in 1,000. The author said he could not reply definitely, but he would expect a battery result from the Benkö battery bulk for bulk. Replying to other speakers the author stated that the emf. of a diffusion cell began to fall after 10 hours' use, and that the electrochemical efficiency of the zinc in a Benkö cell was about the same as in an ordinary primary cell.

General Physics and Chemistry.

Chemical Effects of Canal Rays.—It has been the source of a long series of discussions as to whether chemical elements in the form of canal rays will react with other substances with a greater amount of kinetic energy than is possible if brought to reaction otherwise at highest attainable temperatures. V. KOHLSCHUETTER publishes another interesting contribution in the May 15 number of the *Zeitschrift fuer Elektrochemie*. Canal rays were permitted to act upon various substances, the tubes being filled at the start with small amounts (0.01 to 0.04 mm) of either hydrogen, nitrogen, oxygen, argon, carbon dioxide, sulphur dioxide, or hydrogen sulphide. The author found that the products of reaction were identical with those obtained whenever the substance, which had been acted upon by the canal rays, was used as cathode instead. It was impossible to show absolutely that singular chemical reactions could be brought about by the action of canal rays. The chemical reaction observed seemed to be secondary to a primary effect of the rays, possibly a bombardment or a sudden increase in temperature. At all events this primary effect, whatever it may be, is altogether independent of the chemical nature of the gas in the tube. The effect was stronger in the presence of the heavier gases. Compounds such as oxide of copper and sulphide of silver were reduced more readily in nitrogen than in hydrogen. Chloride of mercury was reduced to the metal even though the gas in the tube was oxygen. Many other examples are cited.

Electrical Resistance of Pure Metals at Very Low Temperatures.—For several years Prof. H. KAMERLINGH ONNES, of Holland, has carried out exceedingly interesting experiments on the enormous decrease of electrical resistivity which pure metals undergo at very low temperatures. He has now perfected his experimental facilities for the use of liquid helium (which by the way was found to be an excellent insulator) and the results of his determinations of the resistivity of pure gold and pure mercury at liquid helium temperatures are given in a recent issue of the *Proceedings* of the Koninklijke Akademie van Wetenschappen, Amsterdam, of which a translation in abstract appears in the London *Electrician* of August 4. The author's recent experiments with gold have greatly strengthened his former conclusion that the resistance of pure gold vanishes at helium temperatures. Even more interesting are the experiments with mercury. The measurements were made with the differential galvanometer by the method of overlapping shunts (Kohlrausch) and also by the method of the measurement

of current strength and of potential difference. The value of the mercury resistance used was 172.7 ohms in the liquid condition at 0°c; extrapolation from the melting point to 0°C by means of the temperature coefficient of solid mercury gives a resistance corresponding to this of 39.7 ohms in the solid state. At 43°K (4.3 degrees on the krypton themometer) this had sunk to 0.084 ohms, that is, 0.0021 times the resistance which the solid mercury would have at 0°C. At 3°K. the resistance was found to have fallen below 3 v 10^{-6} ohm, that is, to one-ten-millionth of the value which it would have at 0°C. As the temperature sank further to 1.5°K, this value remained the upper limit of the resistance. The next step was to look for the point at which the resistance first becomes measurable as the temperature is raised. The temperature of this point was found to be slightly more than 4.2°K, at which the resistance was found to be 230 microhms or one-hundred-thousandth of the resistance (solid) at 0°C.

Recent Metallurgical Patents.

Lead and Copper.

Hydrocarbon Blast Furnace.—The use of oil as fuel for producing the necessary heat in a blast furnace was the subject of considerable experimenting by the late John W. Nesmith, of the Colorado Iron Works Company, of Denver. Recently two patents have been issued, assigned to the company mentioned, based on the experimental work inaugurated by the original inventor.

In Fig. 1 is shown a vertical section of a furnace adapted to carry out the invention. The vertical stack *A* is connected

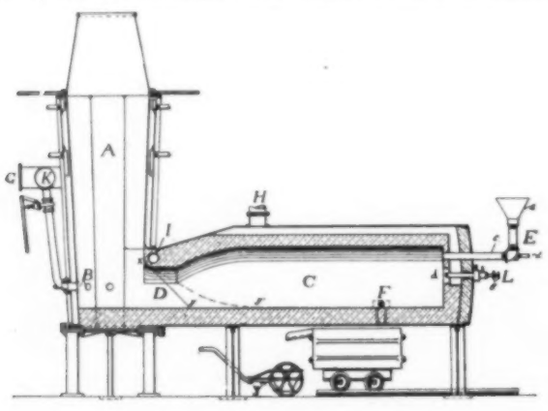


FIG. 1.—HYDROCARBON BLAST FURNACE.

by a passage *D* with a combustion chamber *C*. Tuyeres are located at *B*, and a bustle pipe *K* is connected with an air supply *G*. Air for the combustion chamber is admitted at *H* and passes along the top of the combustion chamber to the inlet shown at *D*. *L* represents an oil burner, *e* being the controlling valve. Provision is made to tap slag and matte at *F*. In order to prevent the burning out of the arch at the connection between stack and combustion chamber a water pipe *I* is located at this point. A hopper with device for forcing fine ore or concentrate into the combustion chamber is located at *E*.

In operation, the regular charge of coarse ore is fed to the stack in the usual manner, and the base of the column fills the opening *D*, the ore taking the natural slope of such material, say 45 degrees. Combustion is maintained in the combustion chamber by means of oil and air admitted at *d*, and the charge gradually smelts to a fluid condition, running onto the hearth of the chamber. The tuyeres are located below the top line of the combustion chamber, as it was found in practice that when located higher on the shaft there was tendency to form a secondary smelting zone in the shaft, causing some of the metals to liquate and flow to the hearth, leaving a skeleton hanging in the shaft. The hot gases from the combustion chamber pass upward through the charge in the shaft and escape.

The combustion chamber is relatively large, as it was found in actual operation that when using petroleum as fuel the volume of space in the chamber should be not less than 35 cu. ft. for each pound of combustible burned per minute.

The provision for introducing fine ore, concentrates or flue dust directly into the combustion chamber offers advantages in smelting such materials, which are always difficult to handle economically. Such fine material is introduced into this furnace from the hopper *E* and accessory devices, by means of a blast of air. Being introduced directly into the combustion chamber they are immediately melted or burned and combined with the molten mass on the hearth. If any part of them is not so smelted it is readily caught in the fusing and semi-fused mass in the stack.

The furnace is thus adapted to the continuous treatment of coarse ore in the usual manner, and the continuous or intermittent treatment of fine material. By this method it is possible to smelt current productions of flue dust, as well as moderate quantities of fine ore and concentrates. Furthermore, in the case of sulphide concentrates, the sulphur contained is utilized as fuel and reduces the quantity of fuel oil required. (995,365-6, June 13, 1911.)

Ore Roasting Furnace.

An annular mechanical roasting furnace adapted for the treatment of pyrite or blende is the basis of a patent of Curt Pfaul, of Blasewitz, Dresden, Germany. A representation of the furnace is given in vertical section in Fig. 2. It has a helical bed with rabbles *b* traveling parallel to it, they being

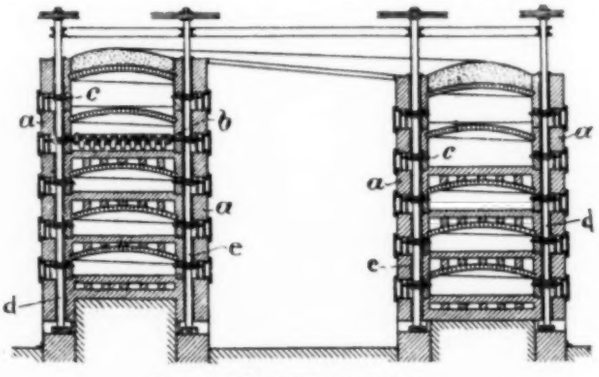


FIG. 2.—ROASTING FURNACE.

carried on rollers *c* bedded in the walls of the furnace. The vertical shafts *d* and *e* upon which the rollers are mounted are arranged at intervals in the circular walls of the furnace. The frame *b* is simultaneously supported by three rollers *c*, and by turning the shafts in either direction the frame carrying the rabbles may be made to move upward or downward in the furnace. The scheme avoids the production of the large quantities of dust formed in the multiple hearth furnaces where the rabbled material falls from one hearth to the other. In this furnace the rabbling is accomplished as the frame moves downward through the furnace. It is then returned to the top by reversing the movement of the shafts, and the operation is repeated (989,868, April 18, 1911).

Fume Condensing Apparatus.

The condensation of the solid and soluble parts of furnace gas by means of water and steam sprays continues to attract the attention of inventors despite the fact that apparatus of this kind has not been usefully applied on a large scale. Paul Kestner, of Lille, France, has patented apparatus for this purpose. He recognizes the fact that dry dust has characteristic repulsion for water, but that if it be preliminarily

dampened it may be readily absorbed in water. He therefore bases his process on first spraying the gas with steam and then cooling it, causing the steam to condense on the solid particles in the fume and dampen them. By subsequently spraying the gas with water he claims that the solids are easily removed. The apparatus is represented in Fig. 3, in which *A* is

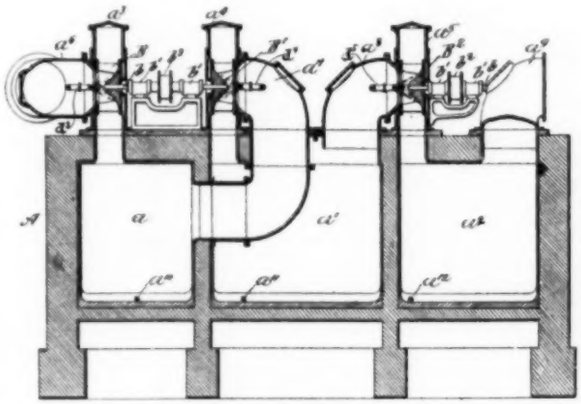


FIG. 3.—KESTNER FUME CONDENSING APPARATUS.

a chamber divided into three compartments *a*, a^1 and a^2: Above these are casings a^3 a^4, a^5 which are connected with their corresponding compartments and in which are located fans, *B*, B^1, B^2 operated by means of pulleys b^2. Spray pipes are arranged to discharge into the face of the fans. As the furnace gases flow into the receptacle at a^6 they are thoroughly mingled

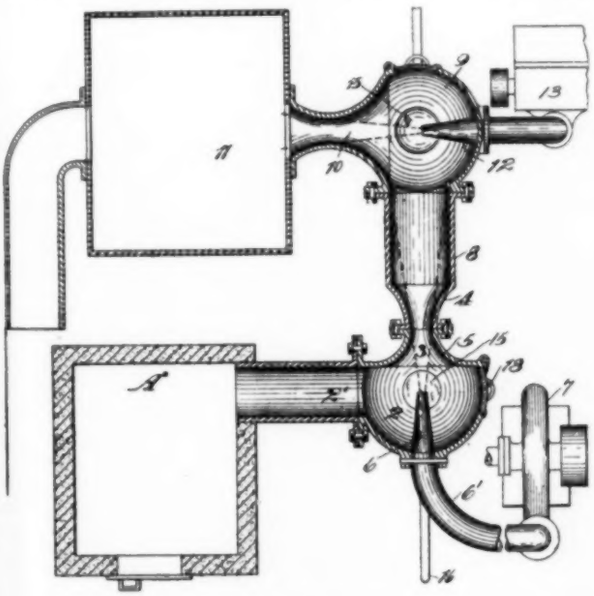


FIG. 4.—FUME CONDENSING APPARATUS.

with steam flowing from the jet against the fan *B*. They then flow into the compartment *a*, where they cool and become wetted with the condensed steam. They then flow sucessfully through the other compartments and fans, being sprayed with water in the latter. The cleaned gases pass out by way of a^5, and the water with the collected fume is accumulated in the chambers *a*, a^1, a^2, from which it flows to other receptacles by way of pipes a^{10}, a^{11}, a^{12} (991,157, May 2, 1911).

Another apparatus for condensing fume in furnace gas from smelters, garbage crematories, etc., is patented by GEORGE C. RICHARDS, of Berkeley, Cal. He first subjects the gases to a jet of water and later to a blast of air. The first jet wets the solid particles and the air blast cools the gases and aids condensation.

The apparatus is shown in plan in Fig. 4. The source of gas is represented by *A*, from which it passes through the passage 2′ to a globular head 2, having a contracted discharge at 3. The water jet is delivered from the nozzle 6. The wetted gas then flows to another globular head 9 where it receives a blast of air from the nozzle 12, which discharges through a contracted orifice 10 into a chamber 11, where the fume is collected. The effect of the angular flow of gas is to break it up and avoid the formation of strata. The accumulations of tarry or other matter in the globular heads may be removed by suitable gates placed in the bottom of each head. The vacuum creating effect of the air blast assists in the force of the induced draft through the apparatus, but its main effect is its cooling action which insures precipitation and condensation (990,009, April 18, 1911).

Electrolytic Production of Oxygen and Hydrogen and Welding and Cutting by the Oxyhydrogen Flame.

The simplest and cheapest way of producing oxygen and hydrogen in equivalent quantities is by electrolysis of water. The process is used in numerous plants in Europe, and the oxygen and hydrogen gases thus produced are widely employed there for welding and cutting by the oxyhydrogen flame and for other purposes. The reason why this country has been behind Europe in this development is probably the fact that the oxy-acetylene flame was early introduced here and was believed, on account of its high temperature, to be superior to the oxyhydrogen flame. It has been since found, however, that both have their distinct fields of applications, and with the increasing demand for hydrogen alone (for instance, in connection with aeronautics) it is but natural that the electrolytic production of hydrogen and oxygen on a large scale has finally found its way to this country.

The International Oxygen Company, of 68 Nassau Street, New York City, has erected a plant at Waverly Park, Newark, N. J., which is one of the most completely equipped of its kind in the world, producing some 10,000 cu. ft. of oxygen daily. This company either furnishes to its customers hydrogen and oxygen gases in cylinders or instals their electrolytic apparatus at the consumers' premises to suit their requirements, so that they produce their own hydrogen and oxygen whenever needed.

The electrolytic cell used is very simple, an outside view being given in Fig. 1 and a diagram in Fig. 2. The iron tank *A* of Fig. 2 serves as the cathode, being connected to the negative pole of the electric supply circuit. From the cover *B* of this tank are suspended the hydraulic joint *C* and a perforated tank *D* which serves as the anode, being connected to the position pole of the supply circuit. It is made of a specially selected low-carbon steel, to prevent the formation of spongey rust.

By means of an asbestos sack *E* suspended from the hydraulic joint *C* two separate compartments are formed, one being the anode compartment and the other the cathode compartment. The cell is thus a true diaphragm cell.

Through the opening *F* in the cover a solution of caustic alkali in distilled water is poured into the hydraulic joint *C* and distributed in the two compartments.

The whole cell is placed on insulating supports *H* of porcelain.

When current is turned on, electrolysis results and the total reaction is the decomposition of water into oxygen gas and hydrogen gas. The oxygen gas is set free at the anode *D* and the hydrogen gas at the cathode *A*. The two gases, thus produced separately from each other, are prevented from mixing by the asbestos sack diaphragm.

The hydraulic joint *C* is a special feature of the cell, as it adjusts its inner working. Another feature is that the oxygen

and hydrogen gases evolved do not pass directly from their compartments to the off-take pipes, but first bubble through water contained in the two lanterns on top of the cell, as shown in Figs. 1 and 2. They enable the operator to see at a glance how the cell is working.

The purity of the gases produced is very high. A sample of hydrogen, produced by this electrolyzer analyzed by the Conservatoire National des Arts et Metier in Paris, showed 99.70 per cent hydrogen, the fraction of the impurities being so small that they were not looked for.

All that is required for the operation of the cell is the daily addition of some distilled water to make up for the quantity decomposed into hydrogen and oxygen. About 1 gal. of water is required to produce 100 cu. ft. of oxygen and 200 cu. ft. of hydrogen. Otherwise the operation of the cells needs no further attention, manipulation or supervision by an expert. As to the electrical energy requirements a joint test made in November, 1910, by the Laboratoire Centrale de l'Electricite and the Conservatoire National des Arts et Metiers with two unit cells of this type of electrolytic cell showed that the production of 1 cu. ft. of oxygen and 2 cu. ft. of hydrogen requires 0.2797

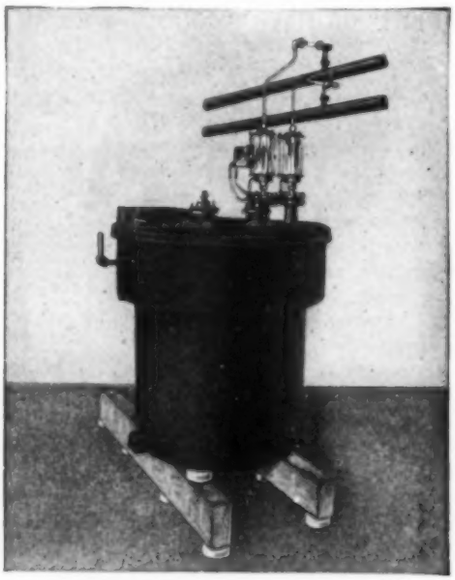


FIG. 1.—VIEW OF CELL FOR ELECTROLYTIC PRODUCTION OF OXYGEN AND HYDROGEN.

kw-hours. Reversely 1 kw-hour produces, 3.54 cu. ft. of oxygen and 7 cu. ft. of hydrogen.

Each unit cell requires a little above 2 volts and from 300 to 400 amp. A current of 350 amp produces about 65 cu. ft. of oxygen and 130 cu. ft. of hydrogen per day.

A plant made up of such unit cells is very flexible, since the units can be arranged in single file and double file, also in many rows and tiers, or in any way to suit the given conditions. Fig. 3 shows three double rows of cells, as installed at the Waverly Park plant of the International Oxygen Company.

The main item of the cost of operation is the cost for the electric current. In New York City the wholesale rates vary from 5 to 3 cents per kw-hour, according to the size of the plant; since 1 kw-hour produces about 3 cu. ft. of oxygen and 6 cu. ft. of hydrogen, the electric power cost for producing 1000 cu. ft. of oxygen and at the same time 2000 cu. ft. of hydrogen would be between $16.70 and $10. However, in large manufacturing plants which have their own power houses the cost of energy is much less; thus it is known that at the South Chicago works of the U. S. Steel Corporation the electrical energy supplied to the electric furnace is charged at the rate of half a cent per kw-hour. At this rate the electric power cost for producing 1000 cu. ft. of oxygen and 2000 cu. ft. of hydrogen would be only $1.67.

Besides the electric power cost there are the cost for attendance which is very small as already mentioned, and the cost for the distilled water which must be added to the cells to make up for the decomposed quantities of water. The latter cost may be calculated from the fact that 10 gal. of distilled water are required to produce 1000 cu. ft. of oxygen and 2000 cu. ft. of hydrogen.

As to the relative advantages of the oxyhydrogen and the oxy-acetylene flame for welding, the lower temperature of the oxyhydrogen flame and its easy regulation are of importance. They render the oxyhydrogen flame superior for thin sheets, especially in the form of kitchen materials and liquid containers that are to be enameled, also for welding aluminium, copper, bronze, and like metals. On the other hand, the oxy-acetylene flame is specially suitable for welding heavy pieces. (A most noteworthy example of heavy oxy-acetylene welding may be mentioned here, namely the recent welding of joints on the pipe line of the Central Colorado Power Company. On the part of the pipe line leading from the reservoir at the power house in Boulder Canyon down to the power house the joints were found leaking, and if the pipe line was to be saved it became

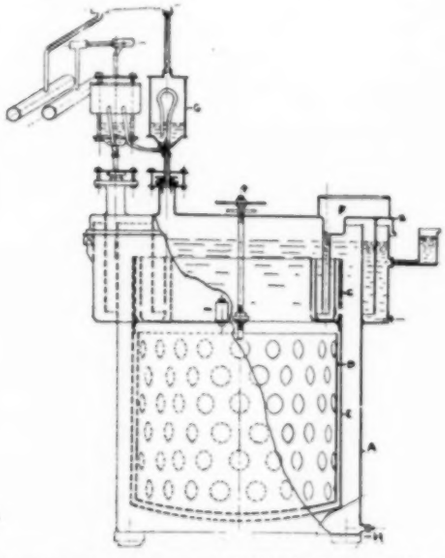


FIG. 2—DIAGRAM OF CELL FOR ELECTROLYTIC PRODUCTION OF OXYGEN AND HYDROGEN.

necessary to weld the joints. This has been carried out by the oxy-acetylene process. It is probably the biggest job of its kind every done and the difficulties were very great, as the pipe line is carried down in a very narrow tunnel and the heat in welding in such a narrow space was intense. Each complete joint took the time of two men for about a day and a half, the method used being quite analogous to that of lead burning. The Central Colorado Power Company must be congratulated on the successful outcome of this exceeding interesting piece of work. The welded pipe line is now giving absolute satisfaction.)

As to oxy-cutting, the following comparison is given by the International Oxygen Company. In oxy-cutting, oxygen and acetylene will cut on the average from 4 to 6 in., whereas with oxygen and hydrogen cuts up to 24 in. have been obtained. The cuts with oxygen and hydrogen are said to be usually smoother and more quickly made than with oxygen and acetylene.

In oxy-cutting the metal is first heated either by the oxy-hydrogen or the oxy-acetylene flame, then the current of hydrogen or acetylene is turned off, and a jet of oxygen is blown upon the molten or melting metal. The cut which results is due to the formation of oxide iron by burning the metal in the

jet of oxygen. This oxide remains outside of the molten steel or iron, and as it is very light, it is blown away by the oxygen under pressure, and thus a cut is produced.

Now it is an important fact that if the oxygen used should contain any appreciable portion of inert gases they will prevent the oxygen from thoroughly oxidizing the steel or iron and will thus render good or deep cutting difficult. The International Oxygen Company attributes the difficulties of oxy-acetylene cutting to the fact that when oxygen combines with acetylene much carbon dioxide is formed which as an inert gas hinders the effectiveness of the oxygen.

The great effect which impurities in oxygen have on the results in cutting were well brought out in a recent paper of J. M. Morehead, of Chicago, before the New York Railroad Club. Numerous tests were made on cutting a mild steel plate ¾ in. thick, the total length of cut being in all cases about 67½ in. In the different experiments oxygen of different purity was

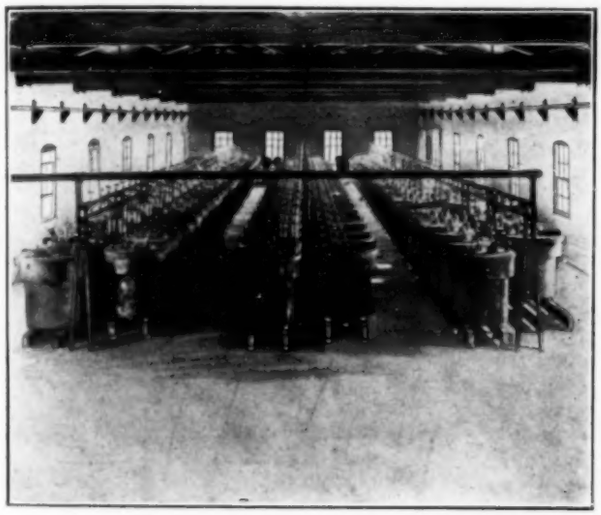


FIG. 3.—CELL ROOM OF INTERNATIONAL OXYGEN COMPANY.

used. We give here only the results of the tests with oxygen of 99.3 per cent, 96.4 per cent and 83.3 per cent purity.

Purity of oxygen, per cent.....	99.3	96.4	83.3
Cutting time in seconds	272	360	855
Oxygen used in cu. ft.	7.5	11.3	18.9

The cut with the 99.3 per cent oxygen was very good, like a hot saw cut. The cut with the 96.4 per cent oxygen was ragged and cindery. The cut with the 88.3 per cent of oxygen was very rough and slaggy, the metal not being properly cut through. The tests show clearly the very appreciable increase in cutting time and quantity of oxygen required, the less the purity of the oxygen. From the tests it would appear that with a percentage of about 80 per cent it is not possible to cut a ¾-in. plate.

An interesting simple application of oxy-cutting is the removal of rivets by the oxy-hydrogen process. An oxy-hydrogen welding torch is used, the rivet is heated until red, the hydrogen is then shut off, and the oxygen burns the rivet away. The meal about the rivet hole remains untouched and the hole is as clean and round as it was before the rivet was put in.

The **Super-Calender** is the title of a new home organ published monthly by the Pusey & Jones Company, of Wilmington, Delaware. Its circulation is limited to paper mill superintendents. The main part will be devoted to questions and answers relating to problems of paper manufacture of general interest. The first issue contains an interesting picture of an early Chinese paper maker and a discussion of two interesting problems. The mechanical make-up is of very high class. Editorially it is of decided originality.

Lining Alloys.

The Lumen Bearing Company, brass founders, of Buffalo, N. Y., and of Toronto, Ont., has sent us an interesting little booklet, entitled "Progress in Lining Alloys," the author of which is the manager of their babbitt department, Mr. Alfred H. Greenburg.

It is emphasized that the term "genuine babbitt" has a rather indefinite meaning as to chemical composition. If it ever meant anything it was simply this: That the preponderant constituent was tin, and that its two other constituents were antimony and copper. Until recent years the term generally implied that the composition was free from lead. This, however, is no longer the case, since the low cost of antimonial lead as a by-product for the last fifteen years and the constant increase in the price of tin have weighed so heavily on the manufacturer of "genuine babbitt" that to-day the term no longer positively excludes lead from its composition. (The composition mentioned in Mr. Isaac Babbitt's first patent in 1839 was 50 parts tin, 5 antimony, and 1 copper. His favorite composition which he later sold to Mr. Phillips, was 10 parts tin, 1 antimony and 1 copper.)

There is no universal bearing composition that is best for all requirements. Hence bearing metal should be specified with great care according to requirements.

The heating of bearings is the principal cause of annoyance, and in cases where the metal punishment is so severe that heating cannot be avoided, a metal of high melting point should be selected. The efficiency of the alloy, therefore, depends upon the quality of the wearing surface that can be produced and maintained under service. A properly selected metal carefully applied, both as to design and workmanship, produces a bearing which, with proper lubrication, has no metallic contact while running. That is, the journal and its bearing are separated from each other by a film of oil which is maintained in operation. As soon as the movement of the journal is stopped, the film of oil is gradually squeezed out and the metallic surfaces are brought into contact. Therefore in selecting the metals for a bearing they should be sufficiently dissimilar so that when starting the machine there will be no danger of scoring the shaft until the oil film shall have been restored.

Aside from the chemical composition, even more important is the method of uniting the different constituents. In general, the metals should be melted in an iron vessel and kept covered as much as possible in order to prevent excessive oxidation. They should be heated considerably above their melting point before using, but must not be kept in a molten state at a high temperature longer than necessary. Overheating should be carefully avoided, and a good rule for general practice is to heat the molten babbitt to a point where it chars a pine stick, at which temperature it casts perfectly.

Further, with respect to analysis. Where the service conditions are severe, owing to great pressure, a metal having considerable compressive strength is necessary regardless of what the speed may be, and this condition would require a relatively high percentage of tin. Where there is high speed and the pressure light and moderate, a metal having a fairly high percentage of lead may be used. In the same manner with intermediate conditions between pressure and speed, correspondingly intermediate composition can be selected.

The surroundings of a bearing and the amount of care and attention which can be given to it must also be taken into consideration. A bearing that is lubricated at long intervals or with a poor grade of lubricant requires a higher grade of metal than that which would be required under more favorable conditions.

The gist of the matter is that while there is nothing very difficult in making babbitt suitable to any clearly specified kind of service, yet it requires an experienced metallurgist.

The little booklet of the Lumen Bearing Company is carefully prepared and well worth reading.

Titanium Rails.

The Titanium Alloy Manufacturing Company, of 1225 Oliver Building, Pittsburg, Pa., have just issued a handsomely illustrated pamphlet, entitled "Facts, Figures and Illustrations Proving That Titanium Has Solved the Rail Problem." It gives very interesting figures on the rapid increase of the use of titanium-treated rails in the last few years.

Three years ago titanium-treated rails were merely being tried on two or three roads, while now they are in service on more than fifty steam railways and on the more important trolley traction lines. Over 40,000 tons of titanium rails are now in actual use. The production of titanium rails in 1910 was 195,940 tons.

The beneficial effect of titanium is not that of an alloying element proper, but the titanium acts simply as a purifier. It is a powerful deoxidizer and removes part, if not all, of the nitrides as well. It prevents brittleness, hence the carbon content of the rail may be materially increased and should be increased if the full benefit of the titanium treatment be desired. The extra cost of the titanium is only about $2 per gross ton of finished rails.

The pamphlet contains interesting illustrations of the structure of titanium-treated and untreated rails, and illustrations with descriptions on the use of titanium rails on many roads in this country.

The following is a test of open-hearth 85-pound rail made recently by the Lackawanna Steel Company for the Delaware & Hudson Co. This steel was treated with 0.15 percent titanium alloy and tested by the National Tube Co. The elastic limit was 100,800 lb. per sq. in., the ultimate strength 147,200 lb. per sq. in., the elongation 12.50 percent, the reduction 16.29 percent.

The analysis was 0.85 C, 0.132 Si, 0.022 P, 0.032 S, 0.88 Mn, 0.065 Ti.

Manufacture of Ferro-Alloys in England.

The National Smelting Company, Ltd., has recently been formed with a capital of £200,000, which, when in full working order, will be practically the first of its kind in England. It has been formed chiefly for acquiring as a going concern the business and assets of Electric Furnace & Smelters, Limited, a company which was incorporated as a private company in 1910 for the purpose of making experiments in the smelting of ferro-alloys and various ores, and whose offices are at 110 Cannon Street, London. It will also acquire other electric furnace patents.

It is reported that the experiments made by Electric Furnaces, Limited, in the making of ferro-tungsten and other ferro-alloys were highly successfully, yielding a metal free from obnoxious impurities at a low cost For the purpose of demonstrating the success accomplished and ascertained by the experiments, a small factory, with plant capable of turning out about 12 tons of tungsten metal per month, was erected at Luton, and is in active daily operation. Adjoining the factory are the works of the Luton Iron Foundry, a business carried on for some years past by Mr. Archibald Finlay Maclaren, the managing director of Electric Furnaces. The benefit of a contract between Mr. Maclaren and Electric Furnaces, whereby the latter acquire an option to take over the land, business and plant of the Luton Iron Company, has been transferred to this company.

Large quantities of tungsten and other alloys are required by British steel manufacturers, over 900 tons of ferro-tungsten being used yearly in Sheffield alone, while the consumption of other high-priced alloys, such as ferro-vanadium, is at least as great. In the past these alloys have been imported to England from other countries. It is intended that the company shall, at its works in Luton, produce at least 1000 tons of ferro-tungsten ad other ferro-alloys annually. Arrangements have been made whereby the company will be able to obtain a supply of electric power at a very low cost.

During the year 1910 an experimental electric furnace was erected at St. Ives, Cornwall, and about 20 tons of tin were smelted there. The result was highly satisfactory and proved the profitable nature of the process. (See the extensive article by Mr. Harden, published elsewhere in this issue.) It is proposed that this company shall proceed with the carrying on of this process, which is provisionally protected.

The directors have entered into an agreement with Mr. Maclaren whereby he becomes the managing director of this company for a period of ten years, and Mr. Johannes Harden will be the technical manager for the like period of ten years. The Board will consist of Mr. Archibald Finlay Maclaren, managing director of Electric Furnaces, Limited, who has been engaged in the foundry business all his life; Mr. Francis Barkley Henderson, director of The National Minerals Corporation, Limited; and Earl Russell, chairman of Plenty & Son, Limited.

Efficient Drying of Cement Rock.

The effect of the fineness to which Portland cement clinker is ground upon the physical property of the resulting cement is well understood, but it is not always appreciated that the fineness to which the material is to be ultimately ground depends to a large extent upon the drying of the product as it comes from the gyratories. Furthermore unless all the hygroscopic moisture is removed the clogging of screens and the cutting down of the capacity of fine grinding mills will result. It will, therefore, be seen that the dryers have played an important part in the development of the cement industry, which in 1910 in this country alone amounted to 70,000,000 barrels, as they have made it possible for cement plants to run at their maximum capacity.

In every industry the efficiency of machines performing certain operations has been studied with keen interest during the last few years in connection with the development of some scientific management scheme and in the cement industry the dryer has received particular notice as it was clearly seen that a considerable saving could be made by the use of the proper dryer and the utilization of the hot gases from the kilns.

Of course, the simplest form of machine for drying cement rock, clay, etc., is a single revolving shell equipped with lifting plates on the inside and having a furnace at one end and a fan at the other end, but the heavy expense of upkeep and its low efficiency due to radiation and the high temperature of the stack gases make it absolutely impracticable. The excessive fuel consumption of this type is decreased when the shell is protected by brick work and the hot gases pass under and back through it.

One of the most efficient and generally used types of dryers is the Ruggles-Coles, which consists of two concentric shells of steel rigidly connected and resting on eight bearing wheels and driven by a heavy cast gear located between the two tires.

It is of particular interest to note how the high economy is attained in the Ruggles type of dryer. In most dryers the hottest gases meet the cold material first with the result that the flames are suddenly quenched and a large amount of carbon passes off unconsumed which is the cause of one of the largest losses in efficiency. In the Ruggles type the hot air enters the central flue at the feed end, passes through this and then back through the space between the two shells, thus drying further the material which is advancing toward the other end.

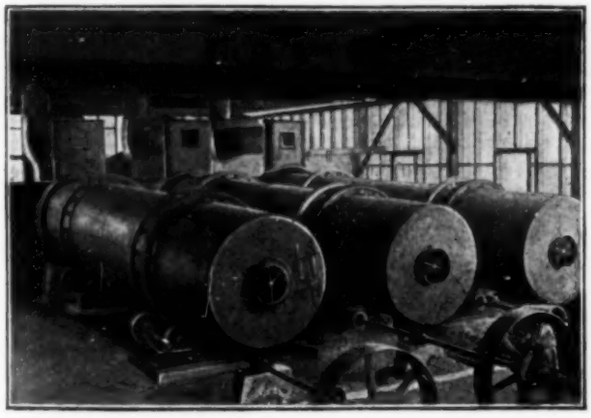
The material to be dryed as it is fed into the space between the concentric shells is picked up by lifting buckets and

dropped onto the inner shell. By revolution of the machine the material is then dropped on the outer shell and this operation continues until it reaches, due to the inclination of the dryer, the discharge end.

It will be seen that the long central flue acts as a large combustion chamber in which space and time are given for the perfect combination of oxygen and monoxide, thereby increasing the efficiency. A resulting exhaust temperature of 100 deg. Fahr. is generally found.

The capacity of any dryer depends directly upon the efficiency and when this is determined the capacity may be figured directly in proportion to the fuel consumed and almost inversely with the moisture contained in the material to be dried. With a stack to produce draft the amount of fuel which can be burned is limited and, therefore, the capacity of the dryer cannot be increased without adding additional stack, which is a troublesome and expensive operation to say the least. By furnishing a fan to run at low speed the Ruggles-Coles dryer may be speeded up by changing a pulley and the capacity of the dryer increased in almost direct proportion. This is a very valuable feature when the material varies in moisture at different seasons of the year and in the cement industry sudden increases of output must be met.

As far as durability is concerned the single-shell type of dryer for cement rock, etc., is practically out of the question as the losses by radiation in such a machine are very large.



DRYER PLANT.

It is not unusual to see these machines red hot for one-third of their length. This means not only constant repairs, but that they are difficult to work near and are sometimes positively dangerous on account of fire. In the two-concentric-shell type, the cold material falling on the top of the inner flue at the hot end, it is impossible for this flue to become overheated, while the outer shell is always practically cold.

As to just what may be expected from a good cement dryer may be ascertained from the following data secured by actual test of a Ruggles-Coles dryer at the Pennsylvania Cement Company, Bath, Pa., in the drying of cement rock.

Calorific value of fuel used	14,100.
Fuel consumed per hour in lbs.	360.
Amount of moisture in material	2.3%
Water evaporated per hour in lbs.	1852.
Water evaporated per lb. of fuel in lbs.	5.1
Material dried per hour in lbs.	78,684.
Fuel per ton of dried material in lbs.	9.2
Heat lost in exhaust air	8.5%
Heat lost by radiation	5.0%
Heat used to evaporate water	40.9%
Heat used to raise temperature of material	45.6%

Hence the total efficiency, defined as the ratio of the heat used to raise the temperature of the material and evaporate the water to the total heat is 86.5 per cent.

Non-Corrosive Bronze.

"Tobin Bronze" is the title of a little booklet issued by the Ansonia Brass & Copper Company of 99 John Street, New York, from which the following information is taken. This bronze is the invention of Mr. John A. Tobin of the Engineer Corps of the U. S. Navy and is manufactured by the Ansonia Brass & Copper Company.

Tobin bronze, a "combination of copper with other metals" combines great mechanical strength with non-corrosibility under difficult conditions.

The specific gravity of tobin bronze is 8.379, hence less than that of copper (8.9).

As to the ability of tobin bronze to withstand chemical influences, tests made by Mr. Charles Taylor of the U. S. Assay Office in New York are interesting. Five samples of tobin bronze, each weighing about 5.5 ounces, were immersed in five different solutions for a period of five months each.

In a saturated salt solution the loss of weight in five months was 0.002 ounces (less than 0.04 per cent). In view of this its wide use in marine construction is not surprising. Thus the hulls of the American cup defenders Columbia and Reliance were entirely made of tobin bronze.

In a caustic potash solution the loss in weight of the 5.5 oz. sample in five months was 0.003 ounces. In a solution of sulphuric acid ("about one-third strength") the loss was 0.014 oz. In a U. S. Standard ammonia solution the loss was 0.161 oz.; in a solution consisting of one-half water and one-half U. S. Standard ammonia the loss was 0.081 oz.

Used in the form of bolts, rods, and plates, its resistance to wear and oxidation makes it a useful material for chemical works, dyewood extract works, tanneries, sugar houses, and coal mines.

When rolled hot, tobin bronze is remarkable for its high elastic limit, tensile strength, hardness, and unifom texture. It can be drawn to a very high degree of tensile strength. But in order to insure the best results in uniformity of tensile strength with the maximum degree of elongation, standard practice of the Ansonia Brass & Copper Company, as to ultimate tensile strength and elongation, of rods of different diameters is as follows:

Diameter of rods.	Ultimate tensile strength.
Up to 1 in.	60,000 lb. per sq. in.
1 to 1½ in.	58,000 " "
1½ to 2 in.	56,000 " "
2 to 2½ in.	54,000 " "
2½ to 3 in.	52,000 " "
Above 3 in.	50,000 " "

The elongation is not less than 30 per cent in 2 inches for all rods 1¼ in. diameter and larger. It is not less than 20 per cent. in 2 in., for all rods smaller than 1¼ in. diameter.

As stated before, the tensile strength can be made higher, but if this is done the elongation is decreased. The elastic limit is about one-half the ultimate tensile strength.

Tobin bronze can be welded electrically.

At a cherry red heat, tobin bronze can be forged as readily on steel. Bolts, nuts and rivets may be made from it either by hand or machinery.

Another feature in working it at cherry red heat is that it can be stamped and drop-forged in the same manner as steel.

Improved Method of Securing Battery Posts.

An arrangement for securing battery posts to the concrete foundation block is described by F. W. Sewell, in the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 3. The arrangement is used at the mills of the Guanajuato Cons. M. & M. Company and the Zacatecas M. & M. Company.

A cast-iron shoe is bolted to the concrete block, and the

shoe, in turn, receives the battery post and holds it in place by suitable bolts and brackets. The shoe weighs 860 lb. and has a bearing surface on the concrete of 4 ft. 6 in. by 1 ft. It has a cavity cast in it 2 ft. long by 10 in. wide and 6 in. deep, into which the battery post fits.

The shoe is secured to the concrete block by two 2½-in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter of the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast-iron washers embedded in the concrete, and are secured thereto by square nuts. Hexagonal nuts or lock nuts are used on the top end to hold down the shoe.

The battery post is held securely in the shoe by means of 1½-in. bolts 2 ft. 7 in. long, connecting brackets on the side of the post with the cast-iron shoe. At each end of the shoe are cast suitable projections which are bored vertically to receive the long bolts mentioned. The two side brackets, which weigh about 100 lb. each, are recessed into the 12-in. faces of the post and held in place by five ⅞-in. rods passing through the post from one bracket to the other. These rods are threaded at both ends, and lock nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of the shoe are two rods, mentioned above, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 1 ft. 2 in. above the top of the shoe.

In erecting the battery, the cavity in the shoe is half filled with oil, and the foot of the post is cut slightly larger than the cavity, the point being cut so that it will just enter the cavity. When the post is forced into place by means of tightening the connecting rods, aided by blows from heavy hammers on top, the oil is forced into the end of the post. This insures protection from rotting due to water. The entire post is then oiled occasionally until no more oil is absorbed, when it is painted.

Fibrous structure is one of the most useful properties of matter, and when found in mineral substance adds greatly to its value. The best grade of asbestos is beautiful silky fiber with a high degree of flexibility, elasticity and tensile strength. It can be spun into a thread so as to run 225 yards to the ounce.

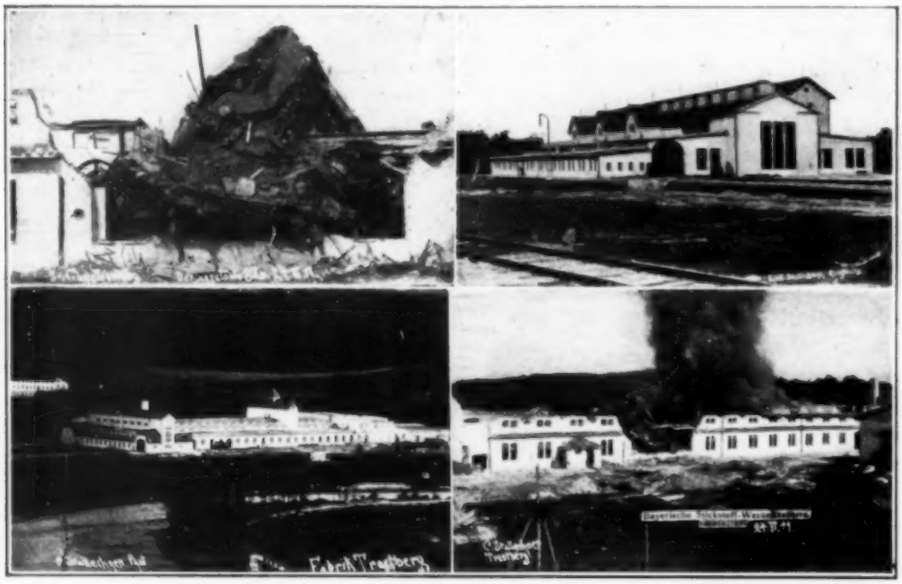
Acetylene Explosion in Cyanamide Plant.

The interesting pictures which are herewith reproduced are taken from some picture postcards sent us from Trostberg in Bavaria, Germany, where the Bayerische Stickstoffwerke have a monster plant, making calcium cyanamide. The factory is located in a valley with the snow-clad Alps surrounding it.

A couple of these pictures are of special interest, showing some details of an acetylene explosion which took place there on the 24th of June this year. Dealing with calcium carbide requires care and precautions, as acetylene is readily formed and even small amounts of acetylene mixed with air will give explosive mixtures. The strength of such an explosion is evident from the pictures.

From time to time reports have come from the European factories concerning similar disasters and it was expected that in this latest constructed plant all precautions had been taken, found in former experience, to make such explosions impossible.

It is only fair to state in this connection that in most older chemical industries where formerly many accidents occurred, these have been very largely reduced in recent years, due to a more careful arrangement and a better construction.

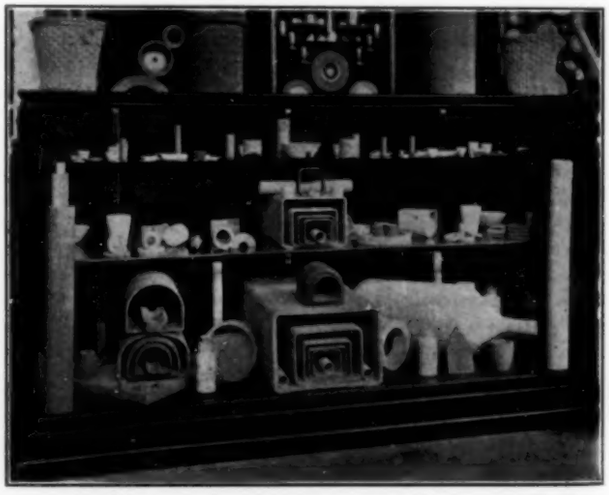


CALCIUM CYANAMIDE PLANT IN BAVARIA.

The lower right-hand picture shows the fire caused by the acetylene explosion; the upper left-hand picture shows a destroyed silo.

Alundum Refractory Exhibit.

The adjoining illustration shows the exhibit of alundum refractories displayed by the manufacturer, the Norton Company, of Worcester, Mass., at the last American Foundrymen's Convention in Pittsburgh.



REFRACTORY EXHIBIT.

As to the properties and manufacture of alundum refractories there is nothing to be added to the exhaustive paper of Mr. L. E. Saunders on page 257 of our May issue.

The exhibit brings out the numerous uses to which alundum refractories are now being put, including muffles of various

shapes and sizes, Cooch type filter crucibles, and crucibles for general laboratory work, combustion boats, extraction thimbles, filter tubes, filtering dishes, electric furnace sections and parts, corrugated furnace cores, arc lamp economizers, tubes, etc. It will be remembered that in the new Hering electric furnace, making use of the pinch phenomenon alundum is used for the electrode tubes with good success.

A New Rotating Anode.

The accompanying illustration shows a new design for a rotating anode for electrolytic analysis, made by Queen & Company, Inc., of Philadelphia, Pa.

The motor is of a new type totally enclosed except on the commutator end which is on top. A fan is mounted on the motor shaft immediately below the contact brush which supplies current to the anode, so that the possibility of gases entering the motor parts is eliminated. In this particular the tripod base, upright and the supporting ring for the dish cathode are heavily enameled. The clamp screws for adjusting the height of the motor and supporting ring have been designed so as not to injure the enamel. Current to the anode is supplied by a specially constructed brush, which bears against a suitable copper ring on the motor shaft. Provision is made to prevent excess of oil from the motor bearings finding its way down to the anode.

ROTATING ANODE.

The tripod base is of iron and heavy enough to prevent vibration and is fitted with soft rubber tips. Binding posts are provided for the current terminals and a rheostat not shown in the illustration controls the motor speed. The illustration does not show the electrodes. The anode is held in position by means of a knurled screw and can be quickly removed.

Institute of Metals.

The annual autumn meeting of the (British) Institute of Metals will be held at Newcastle-on-Tyne on September 20, 21, 22.

Sir Charles A. Parsons is acting as chairman of the local committee and Dr. J. T. Dunn as secretary.

Sessions for the reading and discussion of papers will be held on the mornings of September 20 and 21 at the Armstrong College; on the afternoons of these two days visits to plants will be made. In the evening of September 20 a conversazione will be held, and on September 22 a steamer trip down the Tyne River and back will be made.

Sir Gerard A. Muntz is the president and Mr. G. Shaw Scott, Custom House, Westminster S. W., London, is the secretary of the Institute.

The Non-Ferrous Metal Market.

The markets for non-ferrous metals generally showed a firm tone during August. The copper surplus was reduced in July, and the foreign demand coupled with the reduced production caused some expression of fear that a copper shortage was not outside the bounds of possibility. In zinc the situation is equally favorable, and that metal has a good outlook. Foreign consumption of zinc is heavy, and the price has advanced so that it is not impossible that there may develop a foreign demand for American spelter.

Copper.—The market has been firm, but without advances in price. Lake copper is quoted at 12⅝ to 12¾ cents; electrolytic at 12.40 to 12.45 cents.

Lead.—Business in this market has not been heavy, and the quotations are practically unchanged at 4.50 cents, New York, and 4.42½ to 4.45 cents, St. Louis. The St. Louis quotation is relatively high, as there is usually a greater disparity between New York and St. Louis lead markets.

Tin.—It is reported that the Syndicate in London has temporarily suspended operations, but the foreign market remains firm. The domestic market is good, but there is a scarcity of spot tin owing to dock strikes abroad. August tin was quoted at 42⅝ cents, with spot at 43⅛ cents.

Spelter.—The statistical position is strong and prices have advanced. The latest quotations are 5.90 to 6 cents, New York, and 5.75 to 5.85, St. Louis. The foreign market is reported as firm.

Other Metals.—Aluminium has not changed much during the past month. Quotations stand at 20 to 20½ cents for No. 1 ingots. Sales in antimony are small and prices range from 7.25 to 8.5 cents for various brands. Considerable business has been transacted in quicksilver and prices are steady at $48 to $49, New York, for 75 lb. flasks. San Francisco quotations are $48 for domestic and $45.50 for foreign orders.

Notes.

American Electrochemical Society.—The following gentlemen were elected members at the meeting of the Board of Directors on July 21: Albert A. Heimrod, Newark, N. J.; C. R. Corning, New York City; Frederic S. Mason, New York City; Irving R. Valentine, Schenectady, N. Y.; Thomas D. Robertson, Toronto, Canada; Otto Stalmann, Salt Lake City, Utah; John I. Blair, South Bethlehem, Pa.; Archibald W. Diack, Detroit, Mich.; Harrison W. Johnson, Niagara Falls, N. Y.; John M. Tippett, Colorado Springs, Colo.; Charles R. Barton Ellwood City, Pa.

Vanadium Steel for Aeroplanes.—The July issue of "American Vanadium Facts," published by the American Vanadium Company, of Pittsburgh, contains a great many interesting facts on the use of vanadium in steel making. Thus tubular shells of chrome-vanadium steel, not more than ⅛ inch in thickness are now successfully used as shafts for 50-hp motors to transmit the power on aeroplanes with an ample factor of safety. The shafts weigh 18 pounds each and are 2½ in. outside diameter, 40 in. long, with a 2¼ in. hole bored through their entire length. On account of its excellent forging qualities, its great strength and high elastic limit, and its freedom from crystallization, chrome-vanadium steel finds not only increased use for aeroplane, automobile, and marine motor purposes, but in the construction of chisels, rivet sets, punches, gears, etc.

Oildag for Aeroplanes and Automobiles.—Other interesting examples of the way in which the latest advances in aeroplane and automobile construction have been rendered possible by recent progress in chemistry and metallurgy, are given in a little pamphlet, issued by the International Acheson Graphite Company of Niagara Falls, and entitled "Three Little Stories of Three Big Events and Three Winning Lubricants." Lincoln Beachy, in his recent flight over the Falls of

Niagara, used oildag in the lubricating oil in the engine of his Curtiss biplane. Ray Harroun, the winner of the 500-mile automobile race in Indianapolis, used oildag and gredag. Harry F. Grant, the winner of the big sixth Vanderbilt race on Long Island, used oildag and gredag. It is hardly necessary to add here that oildag is deflocculated graphite diffused in oil (in colloidal condition); gredag is a combination of pure Acheson graphite and a high-grade grease, the two being blended in proper proportions.

The Driver-Harris Wire Company, Harrison, N. J., have developed a new high-fusion wire for combustion furnaces and laboratory heating furnaces. This wire which will be placed on the market under the trade name "Nichrome II" will be efficient for services up to 2000° F. and its chief points of advantage are its extremely high melting point and the proportionately greater resistance to oxidation.

The Electric Furnace Company, Alliance, Ohio, has completed its new furnace-testing laboratory. The building is located on a tract of land secured from the Stark Electric R. R. Co. at their Lake Park power house near Alliance and power for testing purposes will be received direct from the R. R. Company's bus bars. The plant will be supplied with a 200-kilovolt-ampere Westinghouse special regulating transformer, a complete set of electrical recording and indicating instruments, pyrometers, a small chemical laboratory, and equipment for microscopic examination of metals. The Electric Furnace Company has developed a line of electric heating furnaces for forge shops and similar plants, and the new testing laboratory is to enable the company to test new types of furnaces, that are being rapidly called for. The standard type of their electric furnace for heating bars and billets was described in detail and illustrated in a paper by Mr. Thaddeus F. Bailey on page 262 of our May issue.

The first by-product coke ovens in the United States were built in 1893 at Syracuse, N. Y. This country is far behind Germany and other European countries in adopting the economies resulting from coking coal in by-product ovens. Figures compiled by the U. S. Geological Survey show that, while beehive ovens in the United States in 1910 produced coke amounting to only 64.54 per cent of the coal burned, retort ovens in the same year gave a production of 74.9 per cent. The value of the by-products recovered from the retort ovens amounted to about one-third the value of the coke. The by-products recovered were 27,692,858 cu. ft. of surplus gas; 66,303,214 gal. of tar; 70,247,543 lb. of ammonium sulphate or its equivalent; 20,229,421 lb. of anhydrous ammonia; 4,654,282 gal. of ammonia liquor. The value of the recoverable but wasted by-products from beehive ovens in 1910 is estimated by the Survey at from $35,000,000 to $40,000,000. If all the coke produced in the United States were made in retort ovens, there would be a yield of gas over and above that required for the regeneration of the ovens, without by-product recovery, approximating 1,000,000 hp per hour for every day in the year. Complete data on the subject may be obtained from the U. S. Geological Survey, Washington, D. C.

The Bureau of Mines, Washington, D. C., has issued several pamphlets of interest. *Bulletin* 12 treats of the apparatus and methods for the sampling and analysis of furnace gases; *Bulletin* 14 on briquetting tests of lignite, with a chapter on sulphite-pitch binder; *Technical Paper* 1 on the sampling of coal in the mine, and *Technical Paper* 2 on the escape of gas from coal.

Personal.

Mr. A. D. Akin recently has accepted a position as managing engineer for the Monarch Mining & Smelting Company, Yavapai County, Ariz.

Mr. N. H. Brown, of Idaho Springs, Colo., has succeeded Mr. S. B. Tyler as superintendent of mines and mill of the Frontenac Consolidated Mines Company at Central City, Colo.

Prof. F. G. Cottrell, of the University of California, recently went to Montana to confer with Dr. J. L. Holmes of the Bureau of Mines in regard to the smelter smoke problem.

Mr. W. S. Ferris, superintendent of the cyanide plant of the San Rafael y Anexas Company, Pachuca, Mexico, was in Denver recently.

Mr. R. L. Hallett, formerly with the Arizona Smelting Company, Humboldt, Ariz., has been appointed superintendent of the Sandoval Zinc Company's smelter at Sandoval, Ill.

Mr. H. W. Hardinge has gone to Montana on professional business and will return to New York by way of Arizona.

Mr. George J. Hough, chemist and metallurgist, has recently established a laboratory and assay office at Salida, Colo.

Dr. L. D. Ricketts, who was appointed one of a committee of three to investigate the smelter fume problem at Anaconda, Mont., recently spent several days at Anaconda and vicinity.

Mr. Lloyd Robey, superintendent of the cyanide plant of the New York and Honduras Rosario Mining Company, San Juancito, Honduras, has spent a leave of absence in this country.

Mr. Sidney B. Tyler has resigned his position as superintendent of the Frontenac Consolidated Mines Company, Central City, Colo., and has accepted a similar position with the Continental Mining Company, Monclova, Coahuila, Mexico.

Mr. Pope Yeatman, consulting engineer for the Braden Copper Company, Chile, has returned to New York from an inspection of the company's mine and new concentrator.

Obituary.

William L. McLaughlin, of Deadwood, S. D., died at Deadwood on July 28, aged 50 years. Mr. McLaughlin had been identified with the mining and metallurgical industry in a managerial capacity for many years. He was at one time manager of the Horseshoe Mining Company and later manager of the Mogul Mining Company and the Wasp No. 2.

William C. Greene, known for his connection with the copper industry in general, and more particularly with the Green-Cananea copper mines and works at Cananea, Mexico, died at Cananea on August 5. The immediate cause of death was acute pneumonia induced by injuries received a few days ago previous in a runaway accident. Mr. Greene had a rather spectacular rise and fall in financial circles, and ultimately his copper interests were absorbed by other parties.

Mr. Alfred J. Bettles, of Salt Lake City, died in that city on August 3, 1911. Mr. Bettles was one of the foremost metallurgists in the West and had been identified with some of the largest projects in ore dressing. He was associated with Samuel Newhouse during that operator's development of the Cactus and Boston Consolidated companies. The immense mill of the latter was at the time of its construction one of the largest mills in existence. It is designed to treat the low grade copper ores of Bingham, Utah, carrying from 1.5 to 2 per cent. copper. Mr. Bettles adopted radically different steps in the treatment of this ore than were adopted in another mill built about the same time to treat the same ore. His judgment in installing stamps as grinding machines instead of Chilean mills and rolls, was seriously questioned at the time, and the departure attracted much attention in milling circles. The metallurgical results obtained in the Boston mill, however, apparently justified his judgment, although later the mill was remodeled to conform with more advanced practice and to obtain greater tonnage.

Mr. Bettles was recognized as a man of high integrity as well as excellent professional attainments. This combination rendered particularly valuable to his employers and associates at a time when so much depended on both qualities. In the last few years he had given attention to his own interests and to consulting practice. He was fifty-five years of age at the time of his death, which was due to uremic poisoning following an operation.